THE TEXT IS FLY WITHIN THE BOOK ONLY

TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS.

VOL. VIII.

MAY, 1879, TO FEBRUARY, 1880.

EASTON, PA.:
PUBLISHED BY THE INSTITUTE,
AT THE OFFICE OF THE SECRETARY, LAFAYETTE COLLEGE.
1880.

ERRATA.

Page 214, sixteenth and twenty-fifth lines from top, for ferric carbonate read ferrous carbonate.

Page 464, last word, for Ger- read Gei-.

PHILADELPHIA:

SHERMAN & CO., PRINTERS.

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Schirmer, J. F. L.,	1877
Steitz, Augustus,	1876
St. John, I. M.,	1880
Stoelting, Hermann,	1875
Walz, Isidor,	1877
Witherbee, J. G.,	1875
Worthington, Henry R.,	1880

RULES

ADOPTED MAY, 1873. AMENDED MAY, 1875, MAY, 1877, MAY, 1878, and FEBRUARY, 1880.

I.

OBJECTS.

The objects of the AMERICAN INSTITUTE OF MINING ENGINEERS are to promote the Arts and Sciences connected with the economical production of the useful minerals and metals, and the welfare of those employed in these industries, by means of meetings for social intercourse, and the reading and discussion of professional papers, and to circulate, by means of publications among its members and associates, the information thus obtained.

II.

MEMBERSHIP.

The Institute shall consist of Members, Honorary Members, and Associates. Members and Honorary Members shall be professional mining engineers, geologists, metallurgists, or chemists, or persons practically engaged in mining, metallurgy, or metallurgical engineering. Associates shall include all suitable persons desirous of being connected with the Institute and duly elected as hereinafter provided. Each person desirous of becoming a member or associate shall be proposed by at least three members or associates, approved by the Council, and elected by ballot at a regular meeting upon receiving three-fourths of the votes cast, and shall become a member or associate on the payment of his first dues. Each person proposed as an honorary member shall be recommended by at least ten members or associates, approved by the Council and elected by ballot at a regular meeting on receiving nine-tenths of the votes cast; *Provided*, that the number of honorary members shall not exceed twenty. The Council may at any time change the classification of a person elected as associate, so as to make him a member, or *vice versâ*, subject to the approval of the Institute. All members and associates shall be equally entitled to the privileges of membership; *Provided*, that honorary members shall not be entitled to vote or to be members of the Council.

Any member or associate may be stricken from the list on recommendation of the Council, by the vote of three-fourths of the members and associates present at any annual meeting, due notice having been mailed in writing by the Secretary to the said member or associate.

III.

DUES.

The dues of members and associates shall be ten dollars per annum, payable in advance at the annual meeting; *Provided*, that persons elected at the meeting following the annual meeting shall pay eight dollars, and persons elected at the meeting preceding the annnal meeting shall pay four dollars as dues for the current year. Honorary members shall not be liable to dues. Any member or associate may become, by the payment of one hundred dollars at any one time, a life member or associate, and shall not be liable thereafter to annual dues. Any member or associate in arrears may at the discretion of the Council be deprived of the receipt of publications, or stricken from the list of members when in arrears for one year; *Provided*, that he may be restored to membership by the Council on payment of all arrears, or by re-election after an interval of three years.

IV.

OFFICERS.

The affairs of the Institute shall be managed by a Council, consisting of a President, six Vice-Presidents, nine Managers, a Secretary and a Treasurer, who shall be elected from among the members and associates of the Institute at the annual meetings, to hold office as follows:

The President, the Secretary, and the Treasurer for one year (and no person shall be eligible for immediate re-election as President who shall have held that office subsequent to the adoption of these rules, for two consecutive years), the Vice-Presidents for two years, and the Managers for three years; and no Vice-President or Manager shall be eligible for immediate re-election to the same office at the expiration of the term for which he was elected. At each annual meeting a President, three Vice-Presidents, three Managers, a Secretary and a Treasurer shall be elected, and the term of office shall continue until the adjournment of the meeting at which their successors are elected.

The duties of all officers shall be such as usually pertain to their offices, or may be delegated to them by the Council or the Institute; and the Council may in its discretion require bonds to be given by the Treasurer. At each annual meeting the Council shall make a report of proceedings to the Institute together with a financial statement.

Vacancies in the Council may occur by death or resignation; or the Council may by vote of a majority of all its members declare the place of any officer vacant, on his failure for one year, from inability or otherwise, to attend the

Council meetings or perform the duties of his office. All vacancies shall be filled by the appointment of the Council, and any person so appointed shall hold office for the remainder of the term for which his predecessor was elected or appointed; *Provided*, that the said appointment shall not render him ineligible at the next annual meeting.

Five members of the Council shall constitute a quorum; but the Council may appoint an Executive Committee, or business may be transacted at a regularly called meeting of the Council, at which less than a quorum is present, subject to the approval of a majority of the Council, subsequently given in writing to the Secretary, and recorded by him with the minutes.

V.

ELECTIONS.

The annual election shall be conducted as follows: Nominations may be sent in writing to the Secretary, accompanied with the names of the proposers, at any time not less than thirty days before the annual meeting; and the Secretary shall, not less than two weeks before the said meeting, mail to every member or associate (except honorary members), a list of all the nominations for each office so received, stamped with the seal of the Institute, together with a copy of this rule, and the names of the persons ineligible for election to each office. And each member or associate, qualified to vote, may vote, either by striking from or adding to the names of the said list, leaving names not exceeding in number the officers to be elected, or by preparing a new list, signing said altered or prepared ballet with his name, and either mailing it to the Secretary, or presenting it in person at the annual meeting: *Provided*, that no member or associate, in arrears since the last annual meeting, shall be allowed to vote until the said arrears shall have been paid. The ballots shall be received and examined by three Scrutineers, appointed at the annual meeting by the presiding officer; and the persons who shall have received the greatest number of votes for the several offices, shall be declared elected, and the Scrutineers shall so report to the presiding officer. The ballots shall be destroyed, and a list of the elected officers, certified by the Scrutineers, shall be preserved by the Secretary.

VI.

MEETINGS.

The annual meeting of the Institute shall take place on the third Tuesday of February, at which a report of the proceedings of the Institute, and an abstract of the accounts, shall be furnished by the Council. Two other regular meetings of the Institute, shall be held in each year, at such times and places as the Council shall select. Special meetings may be called whenever the Council sees fit; and the Secretary shall call a special meeting on a requisition signed by fifteen or more members. The notices for special meetings shall state the business to be transacted, and no other shall be entertained. All notices may be given by circular, mailed to members and associates, or through the Bulletin, published in the regular organ of the Institute, at the discretion of the Council

Every question which shall come before any meeting of the Institute, shall be decided, unless otherwise provided by these Rules, by the votes of the majority of the members then present. The place of meeting shall be fixed in advance by the Institute, or, in default of such determination, by the Council, and notice of all meetings shall be given by mail, or otherwise, to all members and associates, at least twenty days in advance. Any member or associate may introduce a stranger to any meeting; but the latter shall not take part in the proceedings without the consent of the meeting.

VII.

PAPERS.

The Council shall have power to decide on the propriety of communicating to the Institute any papers which may be received, and they shall be at liberty, when they think it desirable, to direct that any paper read before the Institute, shall be printed in the Transactions. Intimation, when practicable, shall be given at each General Meeting, of the subject of the paper or papers to be read, and of the questions for discussion at the next meeting. The reading of papers shall not be delayed beyond such hour as the presiding officer shall think proper; and the election of members or other business may be adjourned by the presiding officer, to permit the reading and discussion of papers.

The copyright of all papers communicated to, and accepted by the Institute, shall be vested in it, unless otherwise agreed between the Council and the author. The author of each paper read before the Institute shall be entitled to twelve copies, if printed, for his own use, and shall have the right to order any number of copies at the cost of paper and printing, provided said copies are not intended for sale. The Institute is not, as a body, responsible for the statements of fact or opinion, advanced in papers or discussions, at its meetings, and it is understood that papers and discussions should not include matters relating to politics or purely to trade.

VIII.

AMENDMENTS.

These Rules may be amended, at any annual meeting, by a two-thirds vote of the members present, provided that written notice of the proposed amendment shall have been given at a previous meeting.

PROCEEDINGS

OF THE

PITTSBURGH MEETING.

MAY, 1879.

THE hall of the Western Iron and Nail Associations having kindly been placed at the service of the Institute, the opening session was held at 3 o'clock, Tuesday afternoon, May 13th, with an attendance of about one hundred and twenty-five members. Mr. William Metcalf, of Pittsburgh, read an address of welcome on behalf of the Local Committee of Arrangements, to which President E. B. Coxe responded.

The following papers were then read and discussed:

Regenerative Stoves; a Sketch of their History and Notes on their Use, by John M. Hartman, of Philadelphia.

Some Curious Phenomena Observed on Making a Test of a Piece of Bessemer Steel, by William Kent, of Pittsburgh.

On the Use of Determining Slag Densities in Smelting, by Thomas Macfarlane, of Wyandotte, Michigan.

After the reading of these papers each member was presented by the Local Committee with a schedule of excursions, a specially prepared map of Pittsburgh and surroundings, and a directory of industrial works, all tastefully bound, and also with a metal badge bearing the seal of the Institute, to be worn on the excursions.

The second session was held on Tuesday evening.

The following persons, duly proposed for members and associates of the Institute, were, on the recommendation of the Council, unanimously elected. (In this list are included those elected at the final session on Friday morning.)

George Bartol,	Cleveland, Ohio.
Luther S. Bent,	Steelton, Dauphin Co., Pa.
Octave Chanute,	New York City.
Albert H. Childs,	Allegheny, Pa.
Theodore Cooper,	New York City.
Andrew Cosgriff,	Brewster, Putnam Co., N. Y.
William B. Crocker,	New York City.
Henry M. Curry,	Pittsburgh, Pa.
Charles Davis,	Allegheny, Pa.
Henry F. De Bardeleben,	Birmingham, Ala.
Herbert Du Puy,	Pittsburgh, Pa.
Charles L. Fitzhugh,	Pittsburgh, Pa.
W. S. Franklin,	Ashland, Md.
F. G. Fricke,	Pittsburgh, Pa.
Paul A. Fusz,	St. Louis, Mo.
Ignatius Hahn,	Pittsburgh, Pa.
Henry Hargreaves,	South Pittsburgh, Tenn.

Edwin Harrison, . . .	St. Louis, Mo.
William D. Hartupee, . .	Pittsburgh, Pa.
A. T. Hay, ...	Burlington, Iowa.
Frank P. Howe, . . .	Reading, Pa.
Alfred E. Hunt, . . .	Nashua, N. H.
C. C. Hussey, ...	Pittsburgh, Pa.
A. A. Hutchinson, . . .	Pittsburgh, Pa.
Ress James, ...	Johnstown, Pa.
Julian Kennedy, . . .	Braddock, Allegheny Co., Pa.
James Laughlin, Jr., . .	Pittsburgh, Pa.
Alexander J. Leith, . .	Chicago, Ill.
William Lilly, . . .	Mauch Chunk, Pa.
H. F. Mann, ...	Pittsburgh, Pa.
Samuel L. Mather, . . .	Cleveland, O.
P. H. Mell, Jr., . . .	Auburn, Ala.
Orlando Metcalf, . . .	Pittsburgh, Pa.
William Hamilton Merritt, .	St. Catharines, Canada.
George L. Miller, . . .	Latrobe, Pa.
Charles H. Morgan, Jr., . .	Worcester, Mass.
James Morgan, . . .	Pittsburgh, Pa.
T. T. Morrell, ...	Johnstown, Pa.
H. B. Nason, ...	Troy, N. Y.
James W. Neill, . . .	New Orleans, La.
Ferguson G. Parker, . .	Johnstown, Pa.
Richard D. A. Parrott, . .	Greenwood, N. Y.
Charles S. Price, . . .	Johnstown, Pa.
Henry W. Rathbone, . .	Elmira, N. Y.
John Rinard, ...	Braddock, Allegheny Co., Pa.
L. A. Roby, ...	Cleveland, Ohio.
John D. Sanders, . . .	Minè La Motte, Mo.
William Lucien Scaife, . .	Pittsburgh, Pa.
F. Z. Schellenberg, . . .	Irwin's Station, Westmoreland Co., Pa.
Alfred R. C. Selwyn, . .	Montreal, Canada.
James F. Slade, . . .	Troy, N. Y.
John Z. Speer, ...	Pittsburgh, Pa.
G. A. Steiner, ...	Pittsburgh, Pa.
William Swindell, . . .	Allegheny, Pa.
John C. Temple, . . .	Dayton, Ohio.
E. Ray Thompson, . . .	Troy, N. Y.
Henry T. Townsend, . .	Philadelphia, Pa.
E. C. Van Blarcom, . .	Eureka, Nevada.
George Webb, ...	Johnstown, Pa.
William White, Jr., . .	Braddock, Allegheny Co., Pa.
John I. Williams, . . .	Pittsburgh, Pa.
John T. Wilson, . . .	Pittsburgh, Pa.
Ed. L. Wood, ...	Pittsburgh, Pa.
Joseph Wood, ...	Altoona, Pa.
Henry R. Worthington, . .	New York City.
James N. Wright, . . .	Calumet, Michigan.
James B. Young, . . .	Pittsburgh, Pa.
William F. Zimmermann, .	Allegheny, Pa.

ASSOCIATES.

Horace M. Barry,	New York City.
E. V. McCandless,	Pittsburgh, Pa.
T. S. Mathis,	Windham, Colorado.
George F. Milliken,	New York City.
W. B. S. Reed,	Brooklyn, N. Y.
Edgar Richards,	New York City.

The status of the following associates was changed to member:

George B. Cornell,	New York City.
Walter E. Hildreth,	New York City.

The following papers were then read:

Pittsburgh, its Resources and Surroundings, by William P. Shinn, of Pittsburgh.

The Hygiene of Mines, by Dr. R. W. Raymond, of New York.

Accidents in the Comstock Mines, by Professor John A. Church, of New York.

The Tessié Gas Producer, by A. L. Holley, of New York.

Mr. Charles A. Ashburner exhibited and explained, for Professor J. H. Harden, of the University of Pennsylvania, an instrument devised by the latter for ruling equidistant lines, as in cross-section, profile, or topographical drawing, and also for shading cylindrical or irregular curved surfaces with parallel lines drawn at increasing distances from each other, and for general use as a section liner. Illustrations of work done with the instrument were also shown.

Communications were read from the Pittsburgh Chamber of Commerce and the Sportsmen's Association of Western Pennsylvania, extending courtesies, which the Secretary was directed to acknowledge with the thanks of the Institute.

The third session on Wednesday evening was mainly devoted to the continuation of the discussion of Dr. Dudley's papers on Steel Rails, begun at the Baltimore Meeting in February. The discussion was participated in by Messrs. J. W. Cloud, W. R. Jones, R. W. Raymond, A. L. Holley, W. A. Sweet, T. Egleston, E. B. Coxe, William Metcalf, and C. B. Dudley.

Previous to this discussion, Mr. William Metcalf made a short communication on the Swindell Gas Furnace, and Dr. R. W. Raymond described briefly the Salisbury Tar-burning Apparatus.

Mr. G. W. Maynard made a communication on the Thomas and Gilchrist Process of Dephosphorizing Iron, which was discussed by Dr. R. W. Raymond, of New York, who read an account of Snelus's early and successful experiments in the same line; and by Mr. Jacob

Reese, of Pittsburgh, who gave an account of experiments made by him some years ago in making a lime lining for a vessel in which iron ore was reduced.

The final session was held on Friday morning.

President Coxe announced that the Council had, in accordance with a resolution passed at the Baltimore Meeting, appointed the following committee to examine the condition of the collections of the Institute in Memorial Hall, Philadelphia, recently transferred to the Pennsylvania Museum and School of Industrial Art, and report at the next annual meeting of the Institute: Messrs. Egleston, Rand, Blandy, Drinker, and Drown.

The following papers were then read:

The Working of Three Hearths at the Cedar Point Furnace, Port Henry, New York, by T. F. Witherbee, of Port Henry.

An Apparatus for Testing the Resistance of Metals to Repeated Shocks, by William Kent, of Pittsburgh.

Phosphorus in Coal, by A. S. McCreath, of Harrisburg.

Antimony Deposits in Arkansas, by Prof. Charles E. Wait, Rolla, Missouri, read by the Secretary.

Mr. E. B. Coxe exhibited some teeth from an anthracite coal-breaker, which had worn in use, but had retained perfectly their original shape.

Mr. W. E. C. Coxe, Superintendent of the Philadelphia and Reading Railroad rolling-mill, made a brief communication on the wear of one of the iron rails of the company which had carried 67,000,000 gross tons.

Professor Thomas Egleston's paper on the Manufacture of Charcoal in Kilns, and Mr. Thomas Macfarlane's paper on the Classification of the Original Rocks, were read by title.

The following resolutions of thanks for courtesies received were adopted:

Resolved, That the Secretary be directed to express the hearty thanks of the Institute to the railroad companies, the proprietors and superintendents of works, and the various societies and individuals whose graceful courtesy and cordial hospitality have contributed so largely to the pleasure of the members.

Resolved, That the thanks of the Institute are due to the Local Committee, whose thoughtful and admirable arrangements for the profit and pleasure of the visiting members have been beyond all praise.

The President announced that the Council had received a communication from the resident members of the Institute in Montreal, inviting the Institute to hold the autumn meeting of this year in

that city. A second communication had been received, signed by a large number of officials and eminent citizens of Montreal, expressing the wish that this invitation might be accepted. The Council had considered the subject, and had appointed Montreal for the autumn meeting, the precise date of which would be subsequently communicated to the members by the Secretary.

The meeting was then declared adjourned.

EXCURSIONS.

Local Committee of Arrangements.

CHAIRMAN.—William Metcalf.

FINANCE COMMITTEE.—James Park, Jr., William P. Shinn, William Metcalf.

COMMITTEE ON TRANSPORTATION.—William P. Shinn, F. B. Laughlin, H. E. Collins.

COMMITTEE ON REFRESHMENTS.—John H. Ricketson, Thomas M. Carnegie, John L. Gill, Jr.

COMMITTEE ON DIRECTORY.—William Kent, Jos. D. Weeks, John L. Gill, Jr.

SECRETARY AND TREASURER.—Joseph D. Weeks.

The excursions arranged by the Local Committee comprised visits to the most important and interesting industrial works, mines, furnaces, and oil and gas wells in Pittsburgh and vicinity, and were carried out with absolute adherence to the programme. The precision of all the arrangements, the happy selection of places of interest, the ample allowance of time at each place visited, the cordial reception by the proprietors and managers of works, and the thoughtful care exercised by the committee for the entertainment of the members, rendered these excursions memorable in the history of the Institute.

The following is a summary of the schedule of excursions:

Wednesday, May 14th, by steamer Chartiers Valley, on the Ohio and Monongahela rivers: To Davis Island Dam, the works of A. Kloman (Superior Mill), John L. Gill, Jr., Dilworth, Porter & Co., Lewis, Oliver & Phillips, Chess, Smythe & Co., C. G. Hussey & Co., Jones & Laughlin, and Anderson & Co. Before returning to Pittsburgh, the party were taken up the river as far as Glenwood.

Thursday, by train furnished by the Allegheny Valley Railroad Company: The works of Wilson, Walker & Co., the Union Iron Mills, Graff, Bennett & Co., Miller, Metcalf & Parkin, H. K. Porter & Co., Keystone Bridge Works, Lucy Furnace, Standard Oil Refinery and Barrel Works, Brilliant Oil Refinery, Pittsburgh Water-works, Pumping Station and Reservoir, Metcalf, Paul & Co., (Verona Tool Works), Hussey, Howe & Co., Zug & Co., and Mackintosh, Hemphill & Co.

The excursions of this day offering but few attractions for ladies, a special excursion by carriage was arranged for the ladies accompanying the visiting members through Allegheny City, Allegheny Cemetery, and East Liberty Valley.

Friday afternoon, by trains furnished by the Pennsylvania Railroad and Baltimore and Ohio Railroad companies: The Edgar Thomson Steel Works, the United States Iron and Tin Plate Works, and the coal mine of Henry B. Hays & Brother, where a coal-cutting machine was seen in operation.

On Saturday a choice of excursions was offered, one to the Oil Regions of Butler County, and the other to the Connellsville Coke Region. The excursion to the oil regions was made by trains furnished by the Pennsylvania Railroad (West Penn. Division), Parker and Karns City, Karns City and Butler, and Allegheny Valley Railroad companies. The places of interest visited were: Spang, Chalfant & Co.'s Rolling and Pipe Mill, Natural Gas Well and Works of the Carbon-Black Manufacturing Company, and H. L. Taylor & Co.'s Oil Wells at Carbon Centre.

The Connellsville excursion was made by train furnished by the Baltimore and Ohio Railroad Company, the following mines and works being visited: Brown & Cochran's Coke Works at Hickman Run Junction; H. C. Frick & Co.'s Coke Works and Mines at Broad Ford; Charlotte Furnace, and Everson, Macrum & Co.'s Rolling Mill at Everson, and Dunbar Furnace. At Dunbar the party were entertained at dinner by Mr. A. B. De Saules, Vice-President and Manager of the Dunbar Furnace Company. Both excursion parties of this day were returned to Pittsburgh in time to take the evening trains East and West.

On Thursday evening the members were invited by Mr. William P. Shinn, to his house in the suburbs of Pittsburgh, where they were entertained with charming hospitality. An interesting feature of the evening was the presentation to Mr. A. L. Holley, ex-President of the Institute, by Mr. Shinn, on behalf of many friends of Mr. Holley, of a valuable testimonial of their affection and regard.

On Friday evening there was a subscription dinner, at rooms of the Duquesne Club, which had kindly been placed at the service of the Institute, at which over one hundred ladies and gentlemen participated.

PAPERS

OF THE

PITTSBURGH MEETING.

MAY, 1879.

*PITTSBURGH—ITS RESOURCES AND SURROUNDINGS.**

BY WILLIAM P. SHINN, PITTSBURGH, PA.

THE site of Pittsburgh from the first knowledge by white men of its location has been regarded as a position of great strategic importance. One of its earliest visitors, and perhaps the first of whom we have known record, was Washington, who, on November 24th, 1753, stood on the point between the Monongahela and the Allegheny, and looked upon their junction as forming the key to the vast unknown country beyond. On February 17th, 1754, the point was occupied by the erection of a stockade, a forerunner of the fort projected by Washington, but which never saw completion, as in less than three months thereafter the French, with an eye to the dominion of the Mississippi Valley, took possession on April 24th, 1754, of the unfinished stockade, and proceeded to erect Fort Duquesne. In 1755 the expedition, under General Braddock, in its effort to wrest the position from the French, met with its sad defeat, which resulted in the death of its leader. The approximate location of the battle is now occupied by one of Pittsburgh's latest and most successful industrial enterprises, the Edgar Thomson Steel Works.

The French dominion lasted but four years and seven months, and on November 24th, 1758, just five years after Washington projected the fort on this location, the French evacuated Fort Duquesne, and its remains were on the next day taken possession of by General Forbes, who immediately erected Fort Pitt, so called in honor of the Earl of Chatham, then Prime Minister of England, and from January 1st, 1759, the place was known as "Pittsburgh," although it was not until 1764 that a plan of lots was laid out near the fort, and Pittsburgh began to take form. On April 22d, 1794, Pittsburgh was incorporated as a borough, and in 1800 the census gives it 1565 inhabitants. In the interval between the erection of Fort

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* This paper was prepared by request of the Committee of Arrangements. The time at my disposal for its preparation gave but very little opportunity for original research. I am, therefore, indebted for the historical and statistical facts mainly to Mr. James M. Swank's *Iron and Coal in Pennsylvania*, and his statistics of the Iron and Steel Association, Mr. George H. Thurston's *Pittsburgh in the Centennial Year*, and to the assistance of Mr. Joseph D. Weeks, Secretary of the Western Iron and Nail Associations.

Pitt and the year 1800, Pittsburgh had the usual experience of frontier towns, in being besieged by Indians and otherwise harassed.

In the war of 1812 its resources were drawn upon for the arming of Perry's fleet upon Lake Erie, a portion of the cannon having been cast here, and the cordage furnished from rope-works then located at this point; and it is stated that a portion of the cannon and other munitions of war used at the battle of New Orleans were furnished from Pittsburgh. During the War of 1846 with Mexico Pittsburgh was again an important point of rendezvous for troops and for the furnishing of the munitions of war, but it was during the War of the Rebellion that the importance of Pittsburgh to the war-making power of the country was most thoroughly demonstrated. From 1861 to 1864 the Fort Pitt Foundry turned out 2038 cannon and mortars, two of which were the then famous twenty-inch guns known as "Columbiads," the only guns of that size then made in the world. When the rebels had taken possession of the Mississippi River by the fortification of Island No. 10, it was to Pittsburgh that the Government looked for the supply of mortars with which to shell them out—and shelled out they were. In addition to the large number of cannon, the Fort Pitt Works turned out during the same time some 10,000,000 pounds of shot and shell. But "peace hath victories no less renowned than war," and it is to the industrial resources of Pittsburgh rather than to its capacity for destruction that I desire to call your attention at this time.

POPULATION.

The relative rank in population occupied by the city of Pittsburgh has not been commensurate with its industrial importance.

Owing to the fact that the space between the rivers became at an early period in its history undesirable for residences, and partly also to the fact that the coal found in "Coal Hill" on the south side of the Monongahela made that a good location for manufacturing, a large portion of the manufacturing industries of Pittsburgh were located beyond the municipal limits, while a still larger proportion of the residences were to be found in Allegheny City, and in the suburbs of Pittsburgh.

The population of Allegheny County, as shown by the report of the Ninth Census, was at each decade from 1790 to 1870 as follows:

1790,	10,309	1840,	81,235
1800,	15,087	1850,	138,290
1810,	25,317	1860,	178,831
1820,	34,921	1870,	262,204
1830,	50,552		

What it is in 1879 I must leave mainly to conjecture, lest the census of 1880 might materially alter my estimate. But it will certainly be found to exceed 300,000. The population assigned to Pittsburgh in the census of 1870 was 86,076, which made it rank sixteenth in the list of cities of the United States, but Allegheny City had a population of 53,180, and the boroughs on the south side, now a portion of Pittsburgh, had a population of 28,047, so that the population in 1870 of what is now Pittsburgh, was 114,123, which would entitle Pittsburgh as a city, exclusive of Allegheny City, to the rank of twelfth in population in 1870. Allegheny County in 1870 with its population of 262,204 ranked as ninth in population of the counties of the United States, being next below Suffolk County, Mass., which includes Boston, and next above that of Hamilton County, Ohio, which includes Cincinnati.

Pittsburgh is most widely known as a place where passengers stopping for breakfast *en route* between the seaboard and the great West, and looking out at its canopy of smoke express a wonder whether the sun ever shines there, and feel relieved when they are beyond its borders; but little do they know of the economic resources indicated by that smoke. Its sobriquet of the "Iron City" indicates the chief of its many industries, and in order to show that the term is one to which we are entitled, I commence my statement of its resources by reference to

PIG IRON.

The first blast furnace erected in the immediate vicinity of Pittsburgh was located on Two-mile Run, near the present Shadyside Station, on the Pennsylvania Railroad. It was built by George Anshutz about 1792, and was operated only about two years, as its location appears to have been a mistake, based upon the supposition that iron ore was to be found in that vicinity. It was not until 1859 that the second blast furnace, and the first of the existing furnaces in Pittsburgh, was built by Graff, Bennett & Co., on the south side of the Monongahela River, and known as "Clinton" furnace. The pig-iron industry of Pittsburgh is, therefore, less than twenty years old at this date.

There are now twelve blast furnaces in Pittsburgh and its immediate vicinity, and three more in progress. The annual capacity of which, when the three named are completed, will be about 486,000 net tons of 2000 pounds.

The furnaces are in detail as follows:

BUILT.	NAME.	OWNED BY	HEIGHT.	BOSH.	CAPACITY, TONS.
1859	Clinton	Graff, Bennett & Co	45 feet.	12 ft.	14,000
1861	Eliza, No. 1	Laughlins & Co	60 "	17 "	....
1861	Eliza, No. 2	Laughlins & Co	60 "	14 "	70,000
1863	Superior, No. 1	Superior Iron Co	45 "	12 "	....
1868	Superior, No. 2	Superior Iron Co	45 "	12 "	25,000
1865	Shoenberger, No. 1	Shoenberger, Blair & Co	62 "	13 "	....
1865	Shoenberger, No. 2	Shoenberger, Blair & Co	62 "	13 "	48,000
1872	Isabella, No. 1	Isabella Furnace Co	75 "	18 "	....
1872	Isabella, No. 2	Isabella Furnace Co	75 "	20 "	80,000
1872	Soho	Moorhead, McClean & Co	65 "	19 "	31,000
1872	Lucy, No. 1	Lucy Furnace Co	75 "	20 "	....
1877	Lucy, No. 2	Lucy Furnace Co	75 "	20 "	85,000
1879	Furnace A	Edgar Thomson Steel Co. (Limited.)	65 "	13 "	25,000
1879	Furnace B	Edgar Thomson Steel Co. (Limited.)	80 "	20 "	45,000
1879	Furnace C	Edgar Thomson Steel Co. (Limited.)	80 "	20 "	45,000

The following table, compiled from the statistics of the American iron trade as furnished by James M. Swank, Secretary of the American Iron and Steel Association, shows the proportion which the product of pig iron in Pennsylvania bore to that of the United States, and the proportion which the pig iron produced in Allegheny County bore to that of Pennsylvania and the whole United States respectively, during the years 1874 to 1878 inclusive, all stated in tons of 2000 pounds.

DATE.	UNITED STATES.	PENNSYLVANIA.		ALLEGHENY COUNTY.		
	Pig iron made.	Pig iron made.	Per cent. of U.S. product.	Pig iron made.	Per cent. of Pennsylvania product.	Per cent. of U.S. product.
1874	2,689,413	1,213,133	45.11	143,660	11.84	5.34
1875	2,266,581	960,884	42.40	131,856	13.72	5.82
1876	2,093,236	1,009,613	48.23	128,555	12.73	6.14
1877	2,314,585	1,153,356	49.83	141,749	12.29	6.12
1878	2,577,361	1,342,633	52.09	217,299	16.18	8.40

From this statement it will be seen that the pig-iron product of Allegheny County has steadily increased from 5.34 per cent. of the product of the United States in 1874 to 8.40 per cent., or almost exactly one-twelfth, in 1878.

The amount of pig iron brought into Pittsburgh by rail and river during 1878 was 250,476 gross tons,

Equal to	280,533	net tons.
Adding to this the pig iron made in Allegheny County during the year, ...	217,290	"
Makes a total of pig iron made and brought into Allegheny County during the year 1878, .	497,832	"
Or 19.32 per cent. (nearly one-fifth) of the total product of pig iron for the United States. To this add		
Muck bars, ...	2,033	"
Blooms and billets, ...	35,791	"
Scrap iron, ...	64,536	"
And we have a grand total of, ...	600,192	"

of pig iron, muck bars, blooms, billets, and scrap, most of which was consumed in Allegheny County during the year 1878, an amount nearly equal to one-fourth of the whole pig-iron production of the United States for that year.

ROLLED IRON.

In 1811 Wm. B. Foster, Sr., erected upon "Grant's Hill," on High Street, a forge and machinery for manufacturing iron bars as well as some shapes like plow colters. The location is fixed by a deed from Wm. B. Foster to Magnus M. Murray, dated June 27th, 1812, for the undivided half of lots . . . . "on which are erected buildings and machinery for the manufacture of iron."

The first rolling mill in Allegheny County, owned by Christopher Cowan, was built in 1812, and known as the Pittsburgh Rolling Mill; the second, the Union Rolling Mill (not the present mill of that name), was built in 1819, and was accidentally blown up and permanently dismantled in 1829, the machinery being taken to Covington, Ky. This mill had four puddling furnaces, the first in Pittsburgh; it was also the first mill in Pittsburgh to roll bar iron, and was the largest and most extensive mill of the kind in the Western country.

The Grant Hill Iron Works were erected in 1821 by William B. Hays and David Adams, near where the court-house now stands.

The Juniata Iron Works were erected in 1824, and were owned by Dr. Peter Shoenberger. The Sligo mill was erected in 1825, where it now stands, by Robert T. Stewart and John Lyon.

Pig metal for the supply of these mills was mostly brought from the Juniata Valley, which also supplied them with blooms. The Juniata pig iron and blooms were hauled over the Allegheny Mountains to Johnstown, usually on sleds in the winter season, and taken

down the Conemaugh, Kiskiminetas, and Allegheny to Pittsburgh on the spring and fall freshets.

In 1821 Pittsburgh had 8 rolling mills, using 6000 tons of blooms, chiefly from the Juniata Valley, and 1500 tons of pig metal. In 1856 there were in Allegheny County 25 rolling mills and 33 foundries.

The six rolling mills in existence in 1826 employed 281 hands, and made 5230 tons of iron, valued at $559,000, and consumed 561,700 bushels of coal. In 1879 the situation in this regard is as follows:

Number of rolling mills completed, ...	34
Number of rolling mills building (at McKeesport), .	1
Number of common puddling furnaces, . . .	769
Number of Danks puddling furnaces, . . .	11
Number of Siemens puddling furnaces, . . .	10
Total puddling furnaces, ...	790
Number of employés, ...	12,172
Annual capacity in tons, ...	500,000 tons.

The number of single puddling furnaces in rolling mills in the United States (counting a double furnace as equivalent to two single) was, in 1878, 4463, so that the number in Allegheny County was 17.7 per cent. of the number of puddling furnaces in the United States; while the number of rolling mills in the United States being 340, Allegheny County contained 10 per cent. of their number.

The following table shows the amount of rolled iron, including sheets and nail plates, produced in Allegheny County and in the United States during the years 1874 to 1878 stated in tons of 2000 pounds:

TOTAL ROLLED IRON, INCLUDING NAILS.*

YEAR.	UNITED STATES.	ALLEGHENY COUNTY.	
	Rolled iron.	Rolled iron.	Per cent. of U. S. product.
1874	1,110,447	274,625	24.73
1875	1,097,867	239,069	21.78
1876	1,042,101	247,943	23.79
1877	1,144,219	268,486	23.46
1878	1,232,686	282,833	22.93

It will be seen that the proportion of rolled iron, exclusive of rails, made in Allegheny County, has varied from 24.73 per cent. in

* The product given for Allegheny County in Mr. Swank's report includes iron rails, but the only iron rails made in Allegheny County from 1874 to 1878 were of very light patterns, from twelve to twenty pounds, and but very small in quantity.

1874 to 22.93 per cent. in 1878 of the whole product of rolled iron, exclusive of iron rails, made in the United States. The falling off in percentage of the rolled product indicated since 1874 is mainly, if not wholly, owing to the fact that the wages paid puddlers and other rolling-mill expert labor in Pittsburgh are from thirty to fifty per cent. higher than are paid for similar labor in rolling mills east of Pittsburgh.

Six of the rolling mills have connected with them nail mills, having an aggregate of 472 nail machines, which produced in 1874 561,995 kegs of nails of 100 pounds each, and in 1878, 441,013 kegs; the former being 11.45 per cent., and the latter being 10.10 per cent. of the product of the United States; the reason for the reduced proportion being attributable to the same causes as that given for rolled iron.

Allegheny County has many specialties in iron among its manufactures, prominent among which are Messrs. Jones & Laughlin's cold rolled iron, for shafting, piston rods, etc.; Messrs. W. D. Wood & Co.'s planished sheet iron, the only successful rival of the Russia sheet iron; the tin and terne plate works of the United States Iron and Tin Plate Co., and Messrs. Kirkpatrick, Beale & Co., the only manufacturers of that material in the United States; and Andrew Kloman's Universal Mill for the manufacture of weldless eye bars and other bridge material, by a new and highly successful process, applicable to iron, but particularly successful with steel.

The specialties above enumerated are found nowhere else in this country, there being many other specialties manufactured here which are not, however, exclusively products of this county.

STEEL.

Pittsburgh stands pre-eminently at the head of the crucible steel production of this country.

In 1813 there was a steel furnace here owned by Tuper & McKowan, which probably made only blister steel. In 1829 an Englishman, named Broadmeadow, made blister steel at Pittsburgh, and about 1831 made a cast steel of low grade in pots of his own manufacture. His works were located at Bayardstown, now Fifth Ward, near the old Fifth Ward Market House.

Josiah Ankrim & Sons, filemakers, Pittsburgh, are said to have succeeded in making their own steel about 1830. In 1831 Messrs.

Whitmore & Havens successfully produced blister steel at Pittsburgh. In 1833 the firm of G. & J. H. Shoenberger commenced to manufacture blister steel here. About 1840 the firm of Isaac Jones & William Coleman was formed at Pittsburgh, and manufactured blister and spring steel, which business they successfully prosecuted until 1845, when they were succeeded by Jones & Quigg, who built the Pittsburgh Steel Works. In 1846 Coleman & Hailman commenced the manufacture of blister and plough steel, and subsequently made all but first quality cast steel. In 1850 there were in Pennsylvania 13 works with an annual product of 6078 tons, of which 6 works with a capacity of 3278 tons were in Pittsburgh. In 1853 the firm of Singer, Nimick & Co., which had been organized in 1848, was successful in producing the higher grades of cast steel for saw, machinery, and agricultural purposes. In 1859 Messrs. Hussey, Howe & Co. were successful in making crucible cast steel of the best quality; and in 1862 Messrs. Park, Brother & Co., accomplished the same result.

There are now in Allegheny County:

Crucible steel works,	12	
Having Siemens pot furnaces,	33, with 1123	pots.
Coke holes,	223	"
Total,	1346	"

Open-hearth furnaces, 4 completed, 2 erecting; total, 6.

The erection of the Edgar Thomson Steel Works, the only Bessemer steel works in Allegheny County, was commenced by Carnegie, McCandless & Co. in the spring of 1873, and in 1874 the organization was changed in name to the "Edgar Thomson Steel Co., Limited." Their works, located at Bessemer Station, on the Pennsylvania Railroad, and Baltimore and Ohio Railroad, and on the Monongahela River, commenced operations on September 1st, 1875. They have two Bessemer converters, blooming mill, and rail and billet mill, and a five-ton forge hammer. The company is also erecting three blast furnaces on its ground near the steel works. This company, although its works have been in operation less than four years, reached in 1878 the largest output in rails and billets made by any works in the United States in any one year; and the largest output of Bessemer ingots made by any two-converter plant in the world.

The production of the various qualities of steel in the United

States and in Allegheny County respectively, during 1877 and 1878, were as follows, stated in tons of 2000 pounds:

KINDS OF STEEL.	UNITED STATES.		ALLEGHENY COUNTY.			
	Total product.		Total product.		Proportion of U. S. product.	
	1877.	1878.	1877.	1878.	Per cent.	Per cent.
Bessemer steel ingots..	560,587	732,226	73,278	99,344	13.07	13.57
Open-hearth steel ingots...	25,031	36,126	800	1,025	3.20	2.84
Crucible cast steel...........	40 430	42,906	24,747	27,365	61.34	64.95
All other steel................	11,924	8,556	8,323	6,579	70.00	76 90
Bessemer steel rails..........	432,169	550,393	54,635	72,246	12.65	13.13

Of the total product of Bessemer and open-hearth steel ingots in the United States, in 1877, there was made in Allegheny County 12.65 per cent.; and of the total product of the United States in 1878 Allegheny County manufactured 13.06 per cent. Of the total product of crucible, cast, blister, and puddled steel manufactured in the United States in 1877 Allegheny County produced 63.16 per cent.; and of the product of the United States in 1878 Allegheny County produced 66.93 per cent.

The percentage of product in open-hearth steel has heretofore been low, but there have been erected and put in operation since January 1st, 1879, two open-hearth steel furnaces, one by Anderson & Co., and one by Hussey, Howe & Co.; two others are in process of erection by Shoenberger, Blair & Co., and Mr. Thomas S. Blair is about to start his open-hearth furnace at Glenwood, heretofore operated in connection with the manufacture of iron sponge. When the open-hearth furnaces now being erected are completed, Allegheny County will take the lead in open-hearth steel, as she has heretofore done in crucible steel.

Pittsburgh has many specialties in steel manufacture, the most important of which is probably the higher qualities of tool steel, which, during the past ten years, has almost entirely supplanted in the markets of this country the English tool steel, a result reached only after the most patient effort, and in opposition to the most determined prejudice of the manufacturers and users of tool steels.

It is notable also in connection with the Bessemer steel manufacture, that the steel for the cables of the East River Bridge was manufactured here into wire rod by Anderson & Co., and that the steel for the Glasgow Bridge, over the Missouri River, the first all-steel bridge in this country, was manufactured under the "Hay"

process by the Edgar Thomson Steel Company, Limited, and rolled by Hussey, Howe & Co. and Andrew Kloman into the various shapes used in constructing that bridge.

GLASS.

The third in importance of Pittsburgh's manufactures is glass. The first glass house in Pittsburgh is said to have been in operation in 1795, and was located near what is now called "Glass House Riffle," in the Ohio River. It is, however, on record that General James O'Hara and Major Isaac Craig commenced arrangements for the manufacture of glass in 1796 with an eight-pot furnace. In 1807, George Robinson and Edward Ensel commenced the manufacture of flint glass, and in 1808 they were sold out to Messrs. Blakewell & Page; their successors, Messrs. Bakewell, Pears & Co. having carried on the business to the present time.

In 1826, there were in operation in Pittsburgh and vicinity, seven glass works, with a capacity of 27,000 boxes of window glass, and of flint glass to the value of $30,000.

The number of glass manufactories now in Pittsburgh is as follows:

	Furnaces.	Pots.	Hands employed.	Value of annual product.
Table ware,	24	242	1895	$2,000,000
Window glass,	24	234	1200	2,000,000
Flint glass bottles,	8	66	619	420,000
Glass chimneys,	9	90	790	500,000
Green glass bottles,	10	67	944	200,000
Total,	75	699	5448	$6,120,000

The amount of glass manufactured in Allegheny County is about one-half of the glass production of the whole United States.

PETROLEUM.

According to *Stowell's Petroleum Reporter*, the earliest mention of the existence of petroleum in the United States was probably contained in a letter of July 18th, 1627, written by the French missionary, Dalarouche. The locality mentioned is supposed to be near the present town of Cuba, Allegheny County, New York. The earliest mention of petroleum in the State of Pennsylvania appears to be by Charlevoix, in his journal of May, 1721, who speaks of a fountain at the head of the Allegheny, "the water of which is like

oil and has the taste of iron," and was used "to appease all manner of pain." It is also interesting to know that on a map published in 1755, the word "petroleum" appears near the mouth of the present Oil Creek, on the Allegheny River. From this time, mention of the Seneca oil, naphtha, or rock oil, as it was indifferently called, becomes more frequent. In 1843 the means of collection was by absorbing it with blankets from the surface of streams or pools, and pits dug for the purpose on the banks of Oil Creek. Ten or twelve barrels a year was the amount collected by each proprietor, and it was all sold and used for medicinal or remedial purposes.

In 1850, Mr. Samuel Kier, of Pittsburgh, who had collected the oil from some salt wells near Tarentum, on the Allegheny River, and who had bottled it as a medicine, "Kier's Petroleum Liniment," began refining it by distillation, and erected a small refinery in Pittsburgh. It was on Mr. Kier's recommendation that Mr. Smith and his sons were engaged by Drake to drill the first well, and the tools for the drilling were made in Mr. Kier's shop. The earliest of the modern developments of petroleum may therefore be said to have originated in Pittsburgh.

About the middle of June, 1859, Colonel E. L. Drake, of New Haven, Connecticut, began at Titusville, Pennsylvania, the Pioneer well of the Pennsylvania oil regions, and on August 31st, 1859, the well yielded about twenty-five barrels a day, and the production of petroleum commenced in a commercial sense.

The production of crude petroleum for the United States for the years 1877 and 1878 was as follows:

Location.	1877. Barrels.	1878. Barrels.
Pennsylvania oil fields,	13,135,671	15,165,462
West Virginia oil fields,	77,172	250,000
Kentucky and Tennessee,	73,000	75,000
California,	73,000	75,000
Ohio,	36,500	45,000
Total production for the United States,	13,490,171	15,608,462
Average daily production,	39,950	42,763

It will be seen therefore that 98 per cent. of all the crude petroleum produced in the United States during the years 1877 and 1878 was found within the limits of Pennsylvania.

The refining of petroleum in Allegheny County was preceded some three years by the production, on quite a large scale, of coal oil by the

distillation of bituminous coal and shales of this vicinity. Of these establishments the pioneer was the "North American," erected in 1858, followed by the "Lucesco," in the same year, and by the "Aladdin," in 1859. All three of these, however, in 1860 and 1861, abandoned the distillation of coal oil and began refining petroleum; and in 1860 there were within the limits of Allegheny County ten refineries of crude petroleum; in 1861 there were seventeen added; in 1862 nine more were built; and in 1863 fifteen more, so that at the end of 1863 there were fifty-one refineries of crude petroleum in this county.

In 1865 the entire exportation from the United States to foreign ports of refined petroleum was 28,072,018 gallons, while the amount shipped east from Pittsburgh was 25,549,385 gallons.

In 1867 there were fifty-eight refineries in the city of Pittsburgh and suburbs, employing about seven hundred hands, and producing about 60 per cent. of the whole foreign exportation of petroleum. Since that time the number of refineries has considerably decreased, but their capacity has increased. In 1876 there were twenty-nine oil refineries in this vicinity, having one hundred and thirty-eight stills, with a weekly capacity of one hundred and twenty-six thousand three hundred and seventy-one barrels of crude petroleum, or about ninety-five thousand barrels of refined oil, their annual capacity being about four million five hundred and sixty thousand barrels. This is a decrease from the number of refineries in 1866 of one-half, but their capacity for production had increased in the aggregate to three times that of the refineries in 1866.

COAL.

Coal lies at the foundation of the manufacturing supremacy of Pittsburgh. It is to the liberal distribution of this mineral in the surrounding hills, of quality unsurpassed, and almost unequalled for general use, and in quantity almost inexhaustible, as the expression is ordinarily used, located far above the water-level, where mining is cheap and where mills can be located at the pit's mouth, that this vast industry in iron, steel, and glass is due. It is estimated that the bituminous coal field by which Pittsburgh is surrounded, and from which she derives revenue, has fifteen thousand square miles area within the State of Pennsylvania. The Upper or Pittsburgh seam alone has been generally worked, while there are a number of other known seams at lower levels. The mines in the vicinity

of Pittsburgh not only supply its own mills, but contribute largely to the supply of distant manufacturing establishments reached by means of navigation on the Ohio River. Coal was formerly floated down the river in large flat-bottom boats, holding each about fifteen thousand bushels, or about five hundred tons of coal. Since 1850 coal has generally been transported in barges, the movement of which is controlled by tow-boats. A single boat will transport from one hundred thousand to one hundred and thirty thousand bushels, or from four thousand to five thousand tons of coal.

According to Thurston, in 1876 there were in the city of Pittsburgh, and on the Monongahela River, and on the several railroads leading into Pittsburgh, one hundred and fifty-eight collieries, employing sixteen thousand six hundred and twenty-nine men, and mining annually about six million seven hundred thousand tons of coal.

The production of coal in the United States for 1878 has been estimated at 49,130,584 tons. Of this there was produced in Pennsylvania:

Anthracite, ...	17,605,262	tons.
Bituminous, ...	13,500,000	"
Total coal production of Pennsylvania, .	31,105,262	"

being 63.3 per cent. of the whole coal production of the United States.

COKE.

The coal in the vicinity of Connellsville, known as the Connellsville field, is the best adapted for producing coke for smelting purposes of any yet discovered in the United States, and its product is used from Lake Champlain to the Rocky Mountains, wherever smelting is carried on. Not only has it supplanted to a great extent the raw coals of the Mahoning and Shenango valleys in their own strongholds, but its use has been found advantageous as a mixture with anthracite coal in the furnaces of Eastern Pennsylvania and New York. The Connellsville field occupies a space some three miles wide and fifty miles long, the vein being about eleven feet in depth, and yielding some eight to nine feet of workable coal. The coal seam is known as the Pittsburgh Bed of Professor Rogers' Report of the First Geological Survey of Pennsylvania of 1842, and it appears to be geologically an extension of the true Pittsburgh seam before spoken of. This coke yields about 89 to 89½ per cent. of

fixed carbon and about nine per cent. of ash, with usually considerably less than one per cent. of sulphur.

The production of this article has been greatly stimulated within the past ten years. According to the latest information there are now 3668 ovens in the Connellsville region, in addition to which there are about 550 ovens in and about Pittsburgh and along the Pennsylvania, and Pittsburgh, Chicago and St. Louis railways, making a total in the Pittsburgh and Connellsville region of about 4218 ovens, with a daily capacity of 5500 tons of coke.

The amount of coke made and shipped since 1875, according to the estimate of Messrs. H. C. Frick & Co., is as follows, stated in tons of 2000 pounds:

1875,	666,495	tons.
1876,	770,758	"
1877,	869,429	"
In first six months of 1878,	538,856	"

Indicating a probable output of over 1,000,000 tons in 1878, an increase of fully fifty per cent. in the output since 1875.

When the ores of Virginia and Western Virginia are properly developed, their union at Pittsburgh with the Connellsville coke will make this for a generation to come the most prominent pig-iron producing centre in the United States.

LEAD.

The Pennsylvania Lead Company, whose smelting works were erected in 1875 for the purpose of producing lead from the base bullion brought from Colorado, Utah, and California, produced in 1878:

12,508 tons pig lead, valued at,	$1,010,640
874,461 ounces fine silver, valued at,	979,400
4369 ounces fine gold, valued at,	96,118
211 tons copper matte, valued at,	18,592
Total value, product of 1878,	$2,103,710

The total lead product of the United States in 1878 was 81,500 tons, of which the Pennsylvania Lead Company produced nearly one-sixth.

During the four months ending April 30th, 1879, the value of the product of the works has been $1,149,506, the increased value being due to the large amount of silver in the bullion pro-

duced at Leadville. These works are located near North Mansfield, on the Pittsburgh, Chicago and St. Louis Railway, about six miles from this city. They treat base bullion produced in the Western States and Territories, the supply coming principally from Utah, Colorado, Nevada, and Montana, and the amount paid the railroad companies for freight during the first four months of 1879 amounted to $122,675.

White lead is a prominent article of Pittsburgh manufacture. In 1876 there were six white lead factories in operation, using annually about 5000 tons of pig lead, and producing about 200,000 kegs of paint leads of twenty-five pounds each.

COPPER.

There are two copper rolling mills at Pittsburgh, and one copper smelting works. The smelting works were originally established to smelt the product of the famous Cliff mine. The two rolling mills produce copper in its various rolled forms, consuming about 1100 tons of cake and ingot copper, and producing from $600,000 to $700,000 worth of rolled and hammered copper per annum.

PIPE AND TUBING.

There are five establishments engaged in the manufacture of wrought iron pipe and boiler flues within the limits of Allegheny County. The establishment of the National Tube Works Company at McKeesport is probably the largest of the kind in this country. They turn out lap-welded tubes of a diameter as high as fifteen inches, and supply them of any length required for boiler flues.

The works of Spang, Chalfant & Co. are worthy of mention as using for their fuel in puddling the natural gas, brought some seventeen miles from the wells in Butler County. The capacity of production of the five works is equal to 80,000 tons per year.

LOCOMOTIVES.

There are in Pittsburgh and Allegheny City two locomotive works. The Pittsburgh Locomotive and Car Works in Allegheny City, manufacture locomotives of standard size as well as those of smaller type for narrow-gauge roads, while the light locomotive factory of H. K. Porter & Co. manufactures mine engines, switching engines, and other engines of light character.

BRIDGE WORKS.

There are two establishments making their main business the manufacture of iron bridges in Pittsburgh. The Keystone Bridge Company, established in 1860, has a national reputation, being the largest works of the kind in the United States. The St. Louis bridge is a monument of its skill, both in constructing and erecting.

The Iron City Bridge Works of Mr. C. J. Schultz has done some very good work, prominent among which is the bridge across the Allegheny River of the Pittsburgh, Fort Wayne and Chicago Railroad.

Of the numerous minor manufactories, of ploughs and other agricultural implements and machinery, bolts and nuts, spikes and rivets, cast iron pipe, iron and steel wire, chain cables, iron forgings, steam and gas fitting materials, safes, axles, etc., the limits of this paper would not suffice to make even brief individual mention. The excursions arranged by your Committee of Arrangements will furnish you an opportunity to see but superficially the vast industries centred about the head of the Ohio, but if my brief statement of the resources of Pittsburgh and Allegheny County has sufficed to indicate their importance in the industrial development of this country, it will have accomplished its object.

A county which has produced one-twelfth of all the pig iron, and used an amount of pig iron, blooms, and scrap equal to one-fourth of all the pig iron product, and which has produced over two-ninths of all the rolled iron (except rails), over one-eighth of all the Bessemer and open-hearth steel, two-thirds of all the crucible, blister, and puddled steel, one-half of all the glass, one-eighth of all the coal, and one-sixth of all the pig lead produced in the United States in 1878, may well be worthy of the attention of this body, representing the capital, the industry, and especially the metallurgical intelligence of this country; and I leave it to your judgment whether it was not well named by the lamented Lincoln "The State of Allegheny."

THE TESSIÉ GAS PRODUCER.

BY A. L. HOLLEY, C.E., LL.D., NEW YORK CITY.

THOSE who are familiar with working gas furnaces will perhaps admit that the ordinary producer is the least satisfactory feature of the whole system, chiefly by reason of its great waste of fuel, both above and below the fire.

The waste of fuel in cleaning the fire has not, as far as I am aware, been carefully measured; it has been roughly estimated at various works as averaging twenty per cent. to twenty-five per cent. The coke that falls into the ash-pit is largely mixed, if not stuck together with clinker; when this coke is not recovered and utilized (and where coal is cheap it does not pay to recover it), the waste is certainly as high as above estimated.

Cleaning a fire on a grate (especially a fire of dirty coal, like much of our Western coal), by inserting a few temporary grate-bars, then withdrawing the regular grate-bars and letting the whole lower part of the fire tumble into the ash-pit, must necessarily be wasteful of fuel, but it has been found to be the most practical system. The fire under a boiler should be thin, to insure perfect combustion; hence it may be moved from one side to the other, so that the clinker and ashes may be uncovered and removed. But a producer fire should be so thick that carbonic oxide only shall rise from it; a thick fire cannot be thus manipulated.

In several of the French works the ordinary producer without grates is employed; the whole body of the fire rests on the floor of the ash-pit. It did not seem to me that less fuel was wasted here than with grates in cleaning fires. The bottom layer of the fire had to be drawn out with hooks, and there was nothing to prevent much coke from coming away with the clinker and ash.

A producer shaped like a cupola furnace has many advantages, but if grates are employed the waste of coke in cleaning is not lessened.

But the waste at the bottom of the fire is only a part of the waste in the ordinary producer. Mr. Emmerton, chemist to the Joliet Iron and Steel Company, has lately made a number of gas analyses from the producers of that company, worked with impure Illinois

coal and slack. The result shows an astonishing waste of fuel in the form of carbonic acid. Mr. Emmerton admits that his best results are about like the worst he has obtained from some foreign works, but his best are very bad, and I have little doubt that his averages are not very different from averages elsewhere with similar coal. I quote some of them:

Samples taken from the gas stack leading from four producers:

Carbonic acid, ...	9.06	10.07	7.89	9.29
Carbonic oxide, ...	14.32	11.89	16.32	13.55

Samples taken from different producers at different stages of the fire:

	Just charged.	Half hour.	One hour.	End.
Carbonic acid, ...	9.47	8.48	8.96	6.72
Carbonic oxide, ...	12.74	16.53	14.52	16.89

With a strong wind blowing under the producers:

	Just charged.	Half hour.	End.
Carbonic acid, ...	8.28	10.86	7.11
Carbonic oxide, ...	13.02	8.55	16.66

Samples taken from a producer with the damper open five inches. Charged 8.05 A.M.:

	9 A.M.	10 A.M.	11.30 A.M.	1 P.M.
Carbonic acid, ...	7.11	6.33	11.24	10.27
Carbonic oxide, ...	15.65	16 60	7 61	9.04

Mr. Emmerton says the high carbonic acid in the 11.30 sample was found to be due to a large hole in the fire from want of stirring. That in the 1 P.M. sample was probably due to the same cause.

Samples taken from a producer with the damper open only four inches. Charged at 8.40 A.M., and not charged again till 1 P.M.

	9 A.M.	10 A.M.	11 A.M.	12 M.
Carbonic acid, ...	6.53	7.74	6.62	6.14
Carbonic oxide, ...	16.42	14.54	14.82	15.16

Mr. Emmerton remarks: "These results show plainly enough the advantage of running the producers as slowly as possible, and of keeping the fires well poked. I was surprised to find the carbonic

acid lower at the end of a charge than at the beginning. It shows that the heat of a fire has more effect on carbonic acid than its depth, or at all events that the chilling effect of a new charge of coal containing about ten per cent. of moisture hinders the reduction of the carbonic acid formed in the lower part of the fire."

It is impossible to see how this enormous waste is to be prevented in the ordinary producer, because the fire must be comparatively thin. Carbonic acid may so quickly get through a solid fire that not all of it is reduced; it is difficult to prevent holes in the fire, through which, as well as up along the walls, this gas will rise. If the fire were made very thick it would require blast, and blast would so increase the temperature as to make cleaning grates still more difficult and wasteful than it is now.

Besides the waste of coke and carbonic acid in the ordinary producer the work of cleaning the fires is extremely hard and hot; it is therefore expensive if well done.

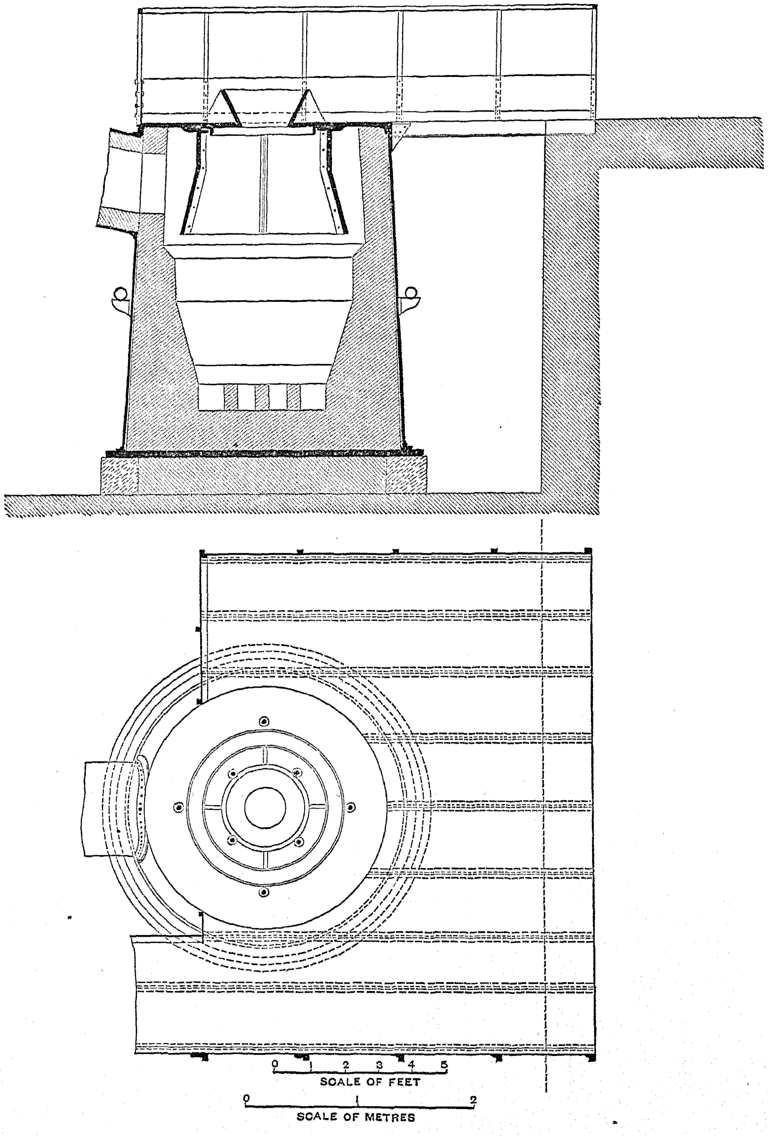
The producer I lately found in extensive use, and superseding all others, at the works of De Wendel, in Lorraine, is called the Tessié Gazogène. Its leading features were devised by that eminent chemist and metallurgist, M. Tessié du Motay, of Paris, although its other features are covered by other patents. In its early form it was like a small blast furnace with some six feet diameter at the boshes and thirteen feet in height.

The drawings show the more improved form, now working some six months with excellent results. The principal dimensions are as follows:

	Feet.	Inches.
Diameter of hearth,	4	3
Depth of hearth,	1	0
Diameter of bosh,	5	2
Diameter of top,	5	10
Diameter of internal ring,	4	0
Internal height,	8	0

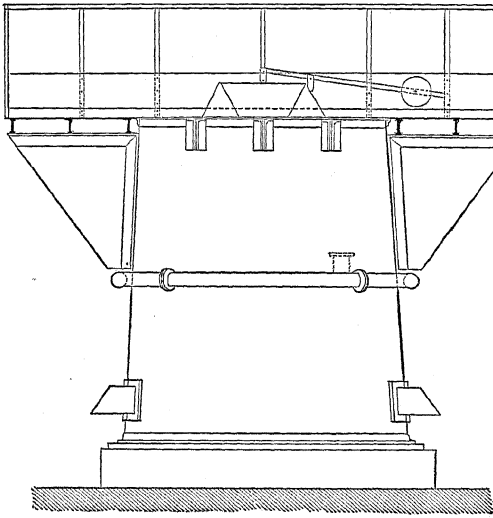
The chief peculiarity and improvement is in the arrangement of the bottom, which is a substitute for a grate. The hearth is cylindrical, and has a brick bottom, in which there are four channels, seven inches deep and six inches wide. These channels extend through the lining and through the shell on both sides, thus forming four holes clear through the hearth, the tops of the channels within the producer being open. A blast-box with a pipe from a common

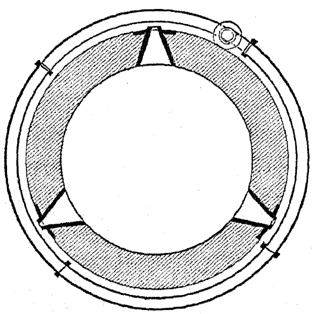
belly-pipe is placed over each end of each channel, so that they all become large tuyeres. When the blast-box covers are removed from the two ends of a channel, a bar may be run clear through the fur-



nace; clinkers and ashes may be readily removed from either end, and the superincumbent coke is held in by the edges of the tuyeres.

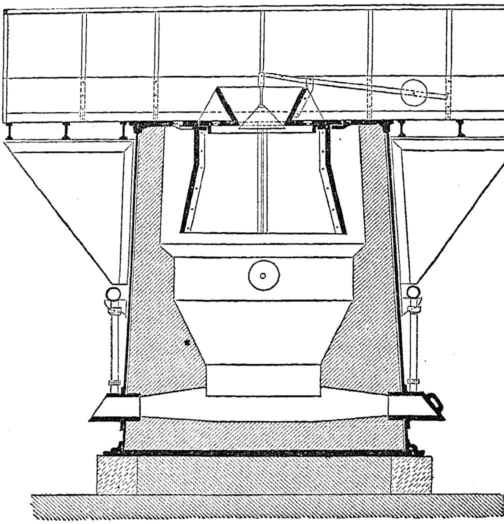
The established working practice and results with these producers I very lately received from Mr. De Wendel.

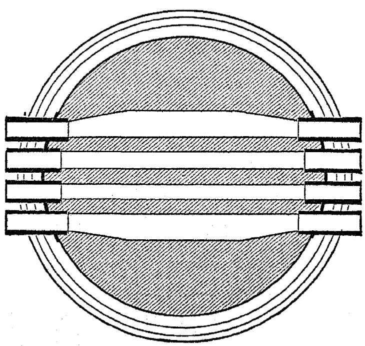




The producer burns 220 pounds per hour of a very inferior coal containing twenty to thirty per cent. of ash. The ordinary producer burns from 150 to 240 pounds of good coal, according to size. The blast pressure is $2\frac{3}{8}$ inches of water; with very bituminous coals some steam is introduced with the blast. The internal ring or bell is kept full of coal, and what poking is required is readily done from the top and side holes. The fire is cleaned every two hours,

once slightly, occupying two minutes, then completely, occupying two men for six minutes. Thus the fire is always kept clean, and hence uniform.





The waste of coke in cleaning has not been definitely ascertained, but it is less than half that in ordinary producers; the working expenses are also about half. Nor has the actual waste of carbonic acid been ascertained. The bricks forming the channels at the bottom of the hearth last indefinitely; no repairs have been as yet made to those at De Wendel's.

The obvious advantages of the Tessié producer are:

1st. Little fuel is wasted in cleaning the fire, chiefly because the ashes and clinker can be pushed straight out of the small tuyere openings without carrying a quantity of coke with them; also because the fire being cleaned often there is no large coherence of coke and clinker. Cleaning the ordinary producer often would aggravate the waste, because a large amount of coke must fall every time the bars are removed.

2d. A very thick fire may be kept at the right temperature by blast to reduce all the carbonic acid before it gets to the top of the bed of fuel, so that its reduction cannot be impeded by fresh, moist coal, as observed by Mr. Emmerton in the ordinary producer.

3d. Even with careless stirring it is difficult to suppose holes in a fire four or five feet deep which will carry up carbonic acid; neither can this gas run up the stepped walls of the producer. In short, with reasonably good attendance, it is difficult to see how any carbonic acid can get through the fire. All the coal should, therefore, be made into combustible gas.

4th. Where the Tessié system is used for making carbonic oxide and hydrogen, to be afterwards carburetted by naphtha for illuminating purposes, it has been found that much hydrogen can be generated, thus displacing nitrogen. The production of hydrogen for metallurgical purposes is the subject of experiments now in progress.

5th. A very important advantage of the blast producer, in addition to the regulation of combustion, is that the gas may be driven down hill or to any distance by the blast without the necessity of cooling-tubes to condense and propel it. The producer may stand conveniently on the general floor, thus avoiding pits and water; and the gas may go hotter to the furnace.

6th. The capacity of a given space and cost of producers is thus largely increased. The ordinary producer must go slow, the blast producer may be urged without danger.

The Tessié producer shown has about eighty per cent. of the internal capacity of the new Terrenoire square producer, but the amount of bad fuel burned in the Tessié is almost exactly the same as the amount of good fuel burned in the Terrenoire. The cupola-shaped producer is the more cheaply and durably built, in a wrought-iron shell or basket.

The cupola producer may obviously be so set as to deliver hot gas to such furnaces as the Ponsard and Swindell, for which air only is regenerated.

THE WORKING OF THREE HEARTHS AT THE CEDAR POINT FURNACE, PORT HENRY, N. Y.

BY T. F. WITHERBEE, PORT HENRY, N. Y.

IN the sections, Figures 1, 2, and 3, are shown three crucibles that have been applied to substantially the same furnace, all the conditions having been the same except a variation of one foot of bosh, and a slight variation in the angle of the same. The bosh of No. 3 was originally the same as No. 2, the additional foot in diameter being the result of three months' cutting.

Beginning with No. 1, which was 5 feet 6 inches diameter, with

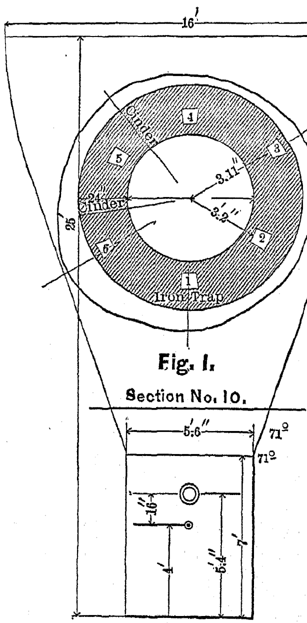


Fig. I.
Section No. 10.

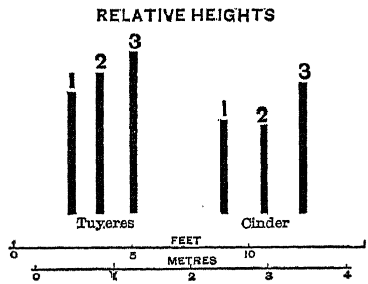
tuyeres 5 feet 4 inches high, and cinder-tap four feet from bottom, great trouble was found in holding iron a proper length of time, four hours being the outside limit when working well, and two hours or less if working badly. "Break-outs" through the iron

notches were not uncommon, alternating with hard notches. This small hearth was built with the expectation of making mill-iron, which would have rapidly enlarged it, and it was of same capacity as those successfully used in our region. Bessemer iron, which we have always made, failed to cut it out, and after two years and sixteen days' use it was not materially enlarged.

The coil tuyeres originally were flush with the brickwork, 5 feet 6 inches from nose to nose, but they were finally drawn back, as shown in Fig. 1, until 6 feet 4 inches apart, and still later Nos. 3 and 6 were further withdrawn, until 7 feet 10 inches apart. The effect of this is shown by the irregular line taken at the point marked section No. 10. It effectually broke up a tendency to hang, which had given some trouble during the blast, in one case lasting $24\frac{1}{2}$ hours, and another 44, followed by slips of 13 feet and 19 feet respectively. Coil tuyeres were used the first three months, and bronze tuyeres ever since, and also the Lürmann front.

The brickwork was only saved by a free use of water, introduced through twelve holes drilled above the tuyeres to within one foot of the nose. Break-outs around the Lürmann front were quite frequent, causing terrific explosions, blowing off the outer course of blocks (bricks), sometimes one-fourth or one-third of the circumference of the hearth.

Fig. 2 shows alterations made for the second blast—the result of a trip through the anthracite regions of Pennsylvania, where hearths of eight feet diameter are not uncommon. Thin walls and



a water-jacket were adopted, the bosh increased to 17 feet, and the angle from 71 to 77 degrees. Owing to the castings coming too large, the hearth was put in 8 feet 8 inches in diameter instead of 8 feet, as intended. The cinder-tap was 4 feet 2 inches high, and tuyeres 6 feet 6 inches; but just before filling a course of 9 inch bricks were laid in the bottom edgewise, making the real height 3

feet $7\frac{1}{2}$ inches and 6 feet $1\frac{1}{2}$ inches, as shown in the accompanying sketch of relative heights.

Blowing-in was very successfully accomplished, which was the only success attained during the blast. To make a long story short, the furnace ran slowly for ten weeks, scaffolded, and at the end of three weeks more had tired out all hands, and exhausted the supply of kerosene oil for miles around us.

Much to our surprise this large hearth held no more iron than the first one, and, safely, not as much. For a time six-hour casts were made by plugging up the cinder-taps and taking the chances, hoping that by holding it in the hearth might enlarge. As a result, a break-out through the Lürmann front occurred, letting about four tons of iron flow into our Kloman cinder-cooler, filling one of the kettles about half full, when the bottom gave way, and the whole arrangement was converted into scrap iron by the explosion that followed.

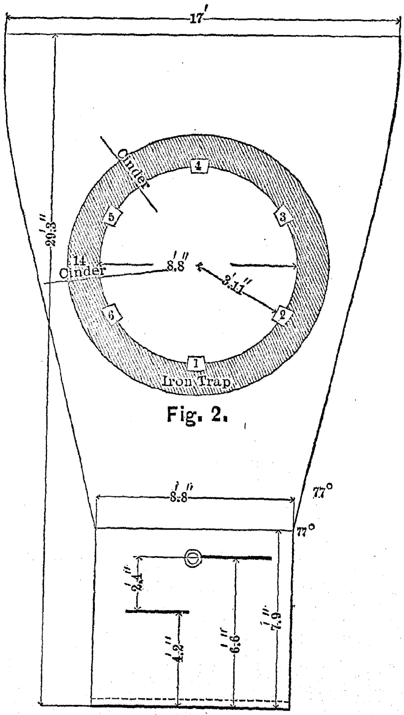


Fig. 2.

The furnace worked with a core, causing a rapid cutting away of the brickwork, which was only limited by the water-jacket and

boiler-iron casing of the boshes. Above the tuyeres the coal was found in direct contact with the water-jacket, and also with the bosh casing over No. 6 tuyere. During the blast the jacket gave no indication of being bare of bricks, though the bosh casing did, and was only saved by water; not a vestige of the bosh remained. Upon removing the bell, a hole 8 feet in diameter was discovered extending down 55 feet, within 9 feet of the tuyeres. Had this been known, and the hole filled up with coal, the furnace might probably have been saved. By the use of kerosene oil we could melt anything in front of the tuyeres, but could not get the mass to run out, owing to its being largely composed of lime.

Probably the reason why this large hearth held so little iron was mainly owing to the great weight of materials in the stack,—over 800 gross tons = 187 lbs. per cubic foot,—the large diameter of hearth and the steepness of the bosh allowing the stock to sit down on the bottom and pack into the crucible.

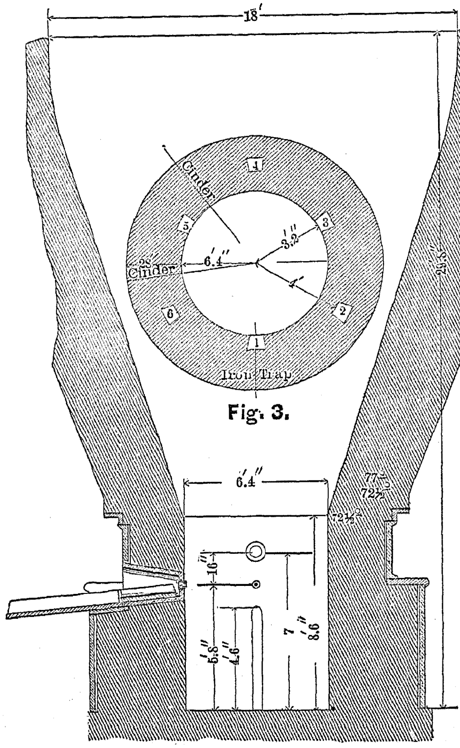
In Fig. 3 you will notice a return to first principles as regards the distance across the tuyeres—6 feet 4 inches, as in No. 1. The tuyeres were placed 7 feet from the bottom, and cinder-tap 5 feet 8 inches. The angle of bosh was changed from 77° to 72½°, not intentionally, but in order to join it where the cutting left the lining in best shape to do so. The iron notch was made continuous, 4½ feet high, as in No. 2, and in blowing-in in each case the crucible was rammed full of sawdust up to cinder-tap. The first tappings of cinder were drawn through the iron notch, which was tapped about half way up. No attempt was made at first to get the bottom of the notch, but it was opened lower and lower each cast, and iron always found. Possibly it might have been found at the bottom much sooner than looked for, but it was thought best to adopt the sure plan of working it down in about two days.

The working of this hearth is very satisfactory, the iron notch has never been hard, and only a few break-outs have occurred. Contrary to the predictions of many there has been no tendency to fill up in the bottom, but on the contrary it has cut down 18 inches during the year it has been in use. Iron has been held ten hours, the regular time being six hours. In only two instances has iron been known to pass through the cinder-tap, and then only in trifling amounts when working too hot, while in Nos. 1 and 2 it caused the loss of many cinder notches. During this blast but two have been used up, while two more were in use 317 days, through which had flowed over 20,000 tons of cinder. They were removed in perfect

condition, but worn thin as paper, and unfortunately melted up by mistake.

In order to use the Lürmann front with best results, the cinder-block must be set high enough, so that iron will neither flow out nor be blown out. Sixteen inches is about the proper distance between the cinder-block and centre of blast tuyeres, which in our case make 7 feet from the bottom of the hearth.

A point in favor of high tuyeres is, that if set high enough, so that there is no probability of the iron rising up to them, the danger of losing them is reduced at least 50 per cent., since cinder will not



cut them, and there is only left the drilling effect of the iron drip on top to contend with. There need be no fear of the crucible filling up; on the contrary, the tendency to cut down will be increased somewhat in proportion to the increased ferrostatic pressure. As the iron notch, cinder-tap, and blast tuyeres have nothing in com-

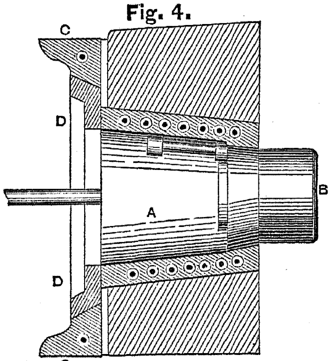
mon, there seems to be no good reason for putting them so nearly on the same level as to interfere with one another.

Fig. 3 shows Nos. 2, 4, and 6 tuyeres withdrawn until eight feet from nose to nose; that is, we have two sets of tuyeres, Nos. 1, 3, and 5, 6 feet 4 inches apart, and the others 8 feet. They were drawn back while in blast to make the furnace drive faster, while those not withdrawn were depended upon to prevent the formation of a core, as in Fig. 2.

Judging by appearances the outer tuyeres do most of the work, naturally keeping wide open, while the inner ones show a tendency to close up about one-half. The result of drawing them back has been favorable, though not as much so as in the case of Nos. 3 and 6 in Fig. 1. Perhaps in the latter case the arch was more effectually broken by withdrawing *opposite* tuyeres than *alternate* ones.

The water-jacket presents no new features except the setting of the blast tuyeres and water-blocks, as shown in section in Fig. 4,

Fig. 4.



where A is the water-block; B, bronze tuyere; C, breast-plate; and D, a plate bolted into end of A, and having a turned joint where in contact with C, making the casing iron and cinder tight at that point, as in fact it is throughout.

LÜRMANN FRONT.

The details of the Lürmann front as applied at Cedar Point are shown in Figs. 5, 6, and 7. E is the arch, a square tapering tuyere, with its inner end closed up so as to take the cinder-notch, F. The arch has a coil of three-quarters inch extra heavy pipe cast in, and also a one-inch pipe, N, surrounding the bronze cinder-notch, en-

tirely independent of the main three-quarter-inch coil. The object of using two pipes is that the arch is very liable to fail around the bronze notch, in which case, if it was protected by a single coil, the whole

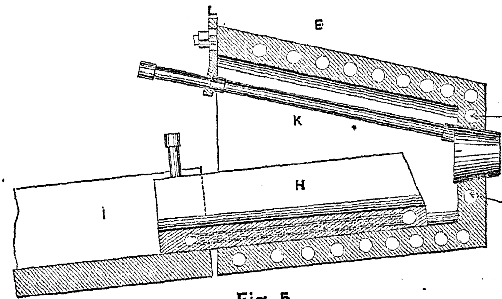
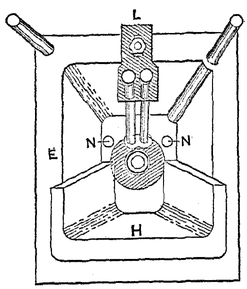


Fig. 5.

arch would have to be renewed. Upon the failure of the pipe, N, in the main casting, the dam-plate is made to go in, a good fit, immediately in front of the inner end, space being left at the end of H to receive it. L is a wrought-iron clamp to hold the cinder-notch



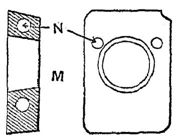
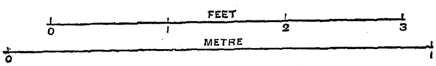


Fig. 7.

Fig. 6.



in place by its water pipes. When the small dam-plate is used it is also held in by its water pipes and similar clamps bolted to the side of the arch. H is a water-cooled spout introduced to save the bottom of the arch, and is the part most frequently destroyed. I is a heavy cast-iron spout to conduct the cinder away, and ought to have a coil cast in it.

In making these castings particular care should be taken to have the coil well covered on the inside of the arch, otherwise the gases

will corrode them in a short time. For the same reason the water pipes of the dam-plate and cinder-notch should be of brass, copper, or galvanized iron, best in the order named.

The coil in the small dam-plate is a ring of one-inch pipe connected by a socket, and cast without any pipe-holes except a small one-quarter-inch pipe at the bottom for a vent, which is finally plugged up. In practice all the pipes and parts are made interchangeable, an extra arch being kept in the machine shop for that purpose.

The pipe-holes in the dam-plate are drilled through guides bolted on to a spare arch when the dam-plate is in position, which insures the proper angle to connect with the leading pipes. If the coils are well covered by cast iron the water-blocks and cinder-arches will probably last two years and over unless destroyed by accident.

To return to the working of the three furnaces: No. 1 was found to work the best above the tuyeres, and No. 3 below, while No. 2 was a failure from top to bottom. It is proper to state that we get the best results from No. 3 if not filled above 12 feet from the top.

If the furnace were to be relined now, we would probably reduce the tunnel-head to 10 feet, and the bosh to 15 or 16 feet, leaving the hearth about as at present. It is very evident that it is much too large for our materials, being rather too much of blast-furnace and storehouse combined.

The performance of the Warwick furnace (Pottstown, Pa.) would seem to make the adoption of its lines advisable. The bell, in each case, was 7½ feet diameter, and is thought to be too large where so much fine ore is used. We have had best results when using all fine ore by putting it in the centre, that is, not spreading it more than 3 feet.

The Bauman double bell was suggested, but data obtained from different parties using it were so conflicting, varying 100 per cent. in size, that it was not considered best to adopt it at present.

The common practice of projecting the tuyeres through the brick-work is not shown in the drawings, not because not approved of, but for the reason that their present location is in the nature of a make-shift to accomplish certain results not connected with the saving of the crucible walls, which the projection of the tuyeres would undoubtedly insure.

THE ANTIMONY DEPOSITS OF ARKANSAS.

BY CHARLES E. WAIT, C.E., M.E., DIRECTOR OF THE SCHOOL OF MINES AND METALLURGY, ROLLA, MISSOURI.

It is said by some that the occurrence of a deposit of sulphide of antimony in Southwestern Arkansas has been known for fifteen or twenty years. Whether or not such is the case I am not prepared to say, but I am not able to find any record of such a discovery, or any analysis of antimony ore obtained from that section, published at so early a period. The present discoveries, that seem to be attracting some attention, date back to the winter of 1873–74. At that time a vein of ore was found by Mr. Robert Wolf, in honor of whom the vein was named, and still is called, "the Wolf lode." At the time of the discovery the ore was unknown to the prospector, and its composition was not thoroughly understood in that locality until samples were sent to several analytical chemists, who pronounced it sulphide of antimony—stibnite.

Since the discovery of this ore there have been three mines opened, and a number of interesting minerals of the antimonial series have been found. I have had an unusually good opportunity to collect specimens from these mines; and it is owing to the constant inquiry concerning the character of these ores that I have collected for publication the results of quite a number of analyses, sufficient I trust to clearly point out their composition. This is one of the few localities in the United States where antimony ore is found in workable quantities; I say workable quantities, because two different shipments of ore have been made to English reduction works, and in both cases excellent returns were made. The discovery of these deposits will, it is hoped, give rise to a prosperous and remunerative industry at no distant day, and when fully developed they may yield ore in sufficient quantity to supply the regulus for the home demand, thus adding wealth not only to those interested, but also to the State of Arkansas.

The three mines are situated in the northern part of Sevier County, and may be reached by the St. Louis, Iron Mountain and Southern Railway to Fulton, thence by wagon-road to Columbus, Nashville, Centre Point, and to the mines. During most seasons of the year the

roads are good, and the distance from Fulton to the mines is about sixty miles.

In the brief sketch to follow I have deemed it best to speak of each mine with its associated ore. The mines are known by the following names, and I give them in the order of their discovery: 1st. Bob Wolf Mine. 2d. Antimony Bluff. 3d. Stewart Lode.

THE BOB WOLF MINE.

This mine is situated about two miles west of the Cosatot River, in section 10, township 7 south, range 31 west. There is a question of doubt as to the exact time of discovery, but it seems to have been in the early part of 1874. Large pieces of the sulphide of antimony were found on the surface, which were weather-worn, but when broken showed the bright metallic lustre peculiar to the freshly-fractured stibnite.

Mining operations on a small scale were commenced without delay, and in a few days a hole was dug from which were taken several tons of very good ore.

The ore is found in a vein nearly a foot in thickness at the surface, with strike N. 13° E., and dip approximately vertical. The veinstone accompanying the ore is quartz, stained with oxide of iron, and has received locally the name of "antimony quartz." The vein has been traced in an easterly direction for a distance of two miles, but at this point the prospect seemed to be the most promising.

The ore appeared in quantity, and several tons of selected ore were taken from the prospect shaft, but upon inquiry it was found that no immediate use could be made of the ore, as there were no smelting works convenient for its reduction. Such being the case they deemed it advisable to cease operations, at least until a market could be found for the mineral.

In January, 1874, several specimens of this ore were sent to this laboratory for examination. A complete analysis of one specimen gave Mr. C. R. Winters the following results:

(No. 1.)	Sulphur,	24.43	per cent.
	Antimony,	57.63	"
	Arsenic,	.34	"
	Bismuth,	4.57	"
	Iron,	.87	"
	Gangue,	11.47	"
	Silver,	trace	"
		99.31	

This was coated with a slight incrustation of a yellowish mineral, supposed to be cervantite. An insufficient quantity, however, was obtained for quantitative analysis.

Another specimen from the same locality was examined, differing little from the above, chiefly in the amount of bismuth and gangue. This gave upon analysis the following:

(No. 2.) Sulphur,	28.52	per cent.
Antimony,	68.58	"
Arsenic,	.49	"
Bismuth,	.51	"
Iron,	.76	"
Gangue,	.95	"
	99.81	

These results were communicated to the American Institute of Mining Engineers, St. Louis Meeting, May, 1874, by Professor Charles P. Williams, to whom I am indebted for the above information. In July of the same year a specimen of the same ore was sent to a chemist in St. Louis for examination. It was reported to be "sulphide of antimony containing about five per cent. siliceous matter, lead and zinc absent, and only traces of iron."

This mine was held by Mr. Wolf only a short time, and was then sold to Messrs. Green & Wells, of Missouri. They sank a small shaft fifty feet west of the original opening, and from it obtained some ore of excellent character. The following is an analysis, made by the author, of ore obtained at the depth of forty-five feet; it is very compact, and of unusual purity:

(No. 3.) Sulphur,	26.59	per cent.
Antimony,	67.38	"
Iron,	.07	"
Silica,	6.13	"
Silver,	trace.	
	100.17	"

The absence of bismuth and arsenic is worthy of mention, since these two metals were found in the specimens taken from the surface. The vein to this depth had assumed a thickness varying from a few inches to one foot.

At the depth of sixty feet another sample was selected for analysis; very compact, and filled with minute crystals of quartz. It was

freed as carefully as possible from impurity, and was examined in this laboratory, under my direction, with the following results:

(No. 4.)	Sulphur,	25.641	per cent.
	Antimony,	66.383	"
	Iron,	.241	"
	Cobalt,	.002	"
	Silica,	7.677	"
	Loss,	.056	"
	Silver,	none.	
		100.00	"

Many other samples have been presented to me for examination, but in no case have there been found any metals in addition to those already mentioned.

The freedom of this ore from lead is rather peculiar, inasmuch as this is decidedly a region of lead-bearing veins, the Davis and Bellah mines being only a few miles distant, while other veins in the immediate vicinity yield lead in small quantities. I have assayed a great many specimens from this mine for gold and silver; in no case have I detected gold, and only traces of silver have been found.

This fact has failed to give encouragement to the owners of the mine in their work of exploration, for they, like all others who own mines in that section, have been daily looking for a change in the character of the ore, *i. e.*, from the ore of one of the common metals to that of silver and gold. Little or no work has been done on this claim, save the sinking of this small shaft, four feet square, to the depth of about sixty-five feet.

THE ANTIMONY BLUFF MINE.

This mine is situated in an easterly direction from the one described, and about two miles distant, on the eastern bank of the Cossatot River, on an elevated bluff in section 6, township 7 south, range 30 west. The deposit was discovered by a man named Batson, but was not given prominence until taken hold of by D. C. Ladd. It is said to be a continuation of the Wolf lode; however, there is some doubt as to the accuracy of this statement. The claim was sold to the Little Rock Mining Company; and in October, 1876, a shaft was commenced upon the vein, which has a strike N. 13° E., with a dip 70° N.

At the surface the vein had a thickness of several inches, but soon increased to two feet, and retained that dimension for about nine

feet in depth. The rock strata through which this vein passes transversely is a very compact siliceous sandstone, while the veinstone accompanying the ore is quartz.

The ore obtained was antimony ochre—cervantite; some pieces were exceedingly pure, weighing as much as three hundred pounds. The following analysis made by the author will clearly show the composition of this beautifully crystallized ochre. Specific gravity = 4.72 dried at 100° C. gave:

(No. 5.)	Antimony,	76.38	per cent.
	Oxygen,	20.03	"
	Sulphur,	.01	"
	Oxide of iron,	.12	"
	Silica,	1.19	"
	Water,	2.23	"
	Silver,	none.	
		99.96	"

The crystals, in many cases, have a bladed structure, resembling somewhat the sulphide, which has, in all probability, been changed to oxide. This conversion is quite evident, as may be seen in several of my cabinet specimens; crystals of from four to five inches in length, and one inch in diameter, are apparently all oxide, but when fractured show a central core one-half inch in diameter of unaltered and mirror-like stibnite.

Another specimen of this mineral was examined in the laboratory of the University of Virginia by Mr. Santos, and reported by Professor J. W. Mallet, to the *Chemical News*, in No. 533. It was compact, with specific gravity = 5.58, and had the following composition:

(No. 6.)	Antimony (diff.),	76.15	per cent.
	Oxygen,	19.85	"
	Water,	3.08	"
	Insoluble residue,	.92	"
		100.00	"

Below this depth, nine feet, the vein diminished in thickness, changed somewhat in direction, conforming more to the rock strata, and yielded ore of ochre and stibnite. At twenty feet in depth the vein was fourteen inches in thickness; below this the ochre had nearly disappeared, and at thirty feet it was replaced by solid glance, the vein here having a thickness of thirty inches. At this point large masses of stibnite, fibrous in structure, were taken out, weighing from one hundred to five hundred pounds.

A specimen of this, specific gravity = 4.49, was examined by Professor Dunnington, University of Virginia, with the following result:

(No. 7.)	Antimony,	71.30	71.8	by formula.
	Sulphur,	28.52	28.2	

By permission I make use of several of his analyses, No. 7, 8, 9, and 10, read before the American Association for the Advancement of Science, Nashville Meeting, August, 1877.

At thirty feet the ore on the north side of the shaft changed somewhat in appearance and indicated the presence of lead.

(No. 8.) A sample of this ore having specific gravity = 5.14 was found to contain lead and antimony in the ratio of 3 to 4, with eight ounces of silver per ton of ore. This is the first specimen of these antimonial ores found to contain lead and silver in weighable amounts, and just at this level and below appear two or three interesting ores, viz.:

(No. 9.) A very compact sulphide of lead and antimony, free from veinstone, specific gravity = 5.33, in all probability zinkenite.

(No. 10.) A compact mass of needlelike crystals, with minute crystals of quartz, specific gravity = 5.19, hardness = 2.5, possessing unusual interest as containing zinc. The analysis shows the following composition:

(No. 10.)	Sulphur,	22.18	per cent.
	Antimony,	32.89	"
	Lead,	36.78	"
	Zinc,	5.07	"
	Iron,	2.62	"
	Silica,	.74	"
		100.28	"

Mr. Dunnington concludes "that this specimen contains jamesonite, in which the iron, with at least an equal amount of zinc, replaces one-third of the lead. While small amounts of zinc have been found with jamesonite, in none of the analyses quoted heretofore does it appear to have entered into that mineral."

The ore taken from this mine is remarkable for its purity, save that obtained from a small vein a few inches in thickness on the north side of the shaft, which yields the minerals containing lead just described.

At the depth of forty feet the vein began to decrease in thickness,

but the mineral possessed no new or striking features. At fifty feet the ore became very fragile, breaking into small crystals and fragments. It was, however, of great purity. A specimen of this ore was examined by myself, the impurities being determined. They occurred in minute quantities only, while the percentage of stibnite present was obtained by difference. The analysis gave:

(No. 11.)	Stibnite,	99.711	per cent.
	Chalcopyrite,	.055	"
	Bismuthinite,	.005	"
	Gangue,	.229	"
	Silver,	none.	
		100.00	"

In the above analysis I have assumed the iron and copper combined with a part of the sulphur as chalcopyrite, and the bismuth to exist as bismuthinite. As to the value of the ore it matters not as to the existing conditions of the metals, for the impurities at best are only trifling. Between the faces of the crystals could be seen with the glass minute yellow stains, due, I presumed, to the presence of copper as chalcopyrite.

At fifty-five feet the mineral decreased in amount, and the vein was divided into several smaller ones, the character of the ore remaining about the same. At this depth blasting became expensive, and for reasons known only to the company mining operations were stopped in June, 1877, and nothing has been done since that time. A specimen of the last ore raised was submitted to analysis and gave the author the following results:

(No. 12.)	Sulphur,	28.51	per cent.
	Antimony,	71.22	"
	Copper,	.05	"
	Bismuth,	.11	"
	Iron,	.24	"
	Silica,	.50	"
	Silver,	trace.	
	Arsenic,	minute trace.	
		100 63	"

This, like the last specimen examined, was well crystallized, the only impurity visible to the eye being small crystals of quartz. During the sinking of this shaft a siliceous slate was encountered in which were imbedded needlelike crystals of arsenopyrite. This is

the only instance in which arsenic has been detected in this mine. All the ores have been free from it, save a trace detected in the last one (12) analyzed.

I have assayed for gold and silver nearly all the ores from this shaft which possessed any peculiarity of lustre, specific gravity, hardness, etc., but have not in any case found the silver to exceed 8 ounces per ton of ore, and gold seems to be absent. A fact worthy of mention is, *all those specimens free from lead are wanting in silver.*

It may be interesting to know that a shipment of ore, obtained from this mine, was made to Messrs. George Hallet & Co., London. Seven tons of selected ore were crushed and shipped in heavy canvas bags. There were four and one-half tons of the sulphide, and two and one-half tons of the antimony ochre. The sulphide was valued at £20 and the ochre at £16 per ton.

THE STEWART LODE.

This is the most extensive deposit of antimony yet found in Arkansas. It is in section 4, township 7 south, range 30 west, in an easterly direction from the Antimony Bluff mine, about two miles distant, and is supposed to be a continuation of that lode. It was discovered by J. H. Anderson in February, 1877. The surface indications in this case were quite interesting. The vein has a strike about N. 13° E., with dip nearly vertical. The ore in large pieces was exposed to view in several places within the distance of a few hundred feet. In many places in this distance the ore and quartz seemed to be a solid mass projecting above the ground.

Soon after this discovery was made mining operations were commenced. The vein was attacked on the surface for several hundred feet, and was removed to the depth of twelve feet. Some fine pieces of ore were taken from this open cut, one piece of apparently solid stibnite, weighing 720 pounds, was shipped to Little Rock; other pieces even heavier were raised to the surface. One other piece furnished 1250 pounds selected ore.

A specimen of this ore was submitted to analysis in the laboratory, and gave the following:

(No. 13.)	Antimony,	69.87	per cent.
	Sulphur,	27.91	"
	Iron,	.02	"
	Zinc,	.01	"
	Silica,	2.69	"
	Silver,	none.	
		100.50	"

At the eastern extremity of this excavation at a depth of 12 feet considerable quantities of very compact ore were found, consisting of granular galena and stibnite intimately associated. A specimen in air-dried condition was examined, and gave me the following:

(No. 14.)	Water,	.03	per cent.
	Lead,	61.12	"
	Antimony,	18 89	"
	Silica,	3.28	"
	Iron,	.06	"
	Sulphur,	16.96	"
	Silver,	.0246	"
		100.36	"

The per cent. value of silver corresponds to 7.17 oz. per ton ore.

(No. 15.) I submitted to assay another specimen consisting of granular galena, from the same locality in the cut, and obtained silver 7.8 oz. per ton ore and a trace of gold. At the western extremity of this excavation was found an exceedingly interesting mineral of a lemon-yellow color, easily crumbling between the fingers, and giving upon charcoal the reactions for lead and antimony; it was pronounced bindheimite.

It was analyzed, and gave the following results:

(No. 16.*)	Oxide of antimony,	49.67	per cent.
	" " lead,	40.35	"
	" " iron,	2.98	"
	Water,	5.98	"
	Silica,	1.14	"
		100.12	"

In order to test the character and extent of ore in this lode, several holes were dug, and in each case satisfactory results were obtained, sufficient at least to warrant a moderate expenditure in sinking a shaft. This shaft, 8 feet by 5 feet, was commenced near the eastern extremity of the cut. Some very large and pure pieces of stibnite were taken from this shaft, as well as one or two interesting minerals.

* I am indebted to Professor Dunnington for this analysis. Read before the American Association for the Advancement of Science.

(No. 17.) One specimen, crystallized, fibrous jamesonite, blue-black color: specific gravity = 5.15, gave—

Sulphur,	22.073	per cent.
Lead,	38.436	"
Antimony,	35.063	"
Iron,	2.531	"
Copper,	.010	"
Bismuth,	.005	"
Cadmium,	.009	"
Silver,	.2229	"
Gold,	trace.	
Silica,	1.576	"
	99.926	"

It will be noticed that this specimen of jamesonite differs somewhat from that given in analysis No. 10, mainly in the presence of copper, bismuth, cadmium in minute quantities, the absence of zinc, and the considerable percentage of silver.

(No. 18.) Another specimen from this mine was recently presented to me; its appearance indicated stibnite with a predominance of jamesonite or zinkenite. It was examined in the laboratory under my direction with the following results, approaching nearly the composition of zinkenite:

Sulphur,	20.55	per cent.
Antimony,	42.37	"
Lead,	27.94	"
Iron,	2.75	"
Zinc,	.15	"
Silica,	5.76	"
Silver,	.048	"
	99.568	"

(No. 19.) Another mineral of much interest was given to me by the former owner of this mine, which resembled somewhat in appearance that given in analysis No. 16. Being desirous to know whether it was identical in composition, I found it to yield upon analysis:

Water,	5.00	per cent.
Oxide of lead,	45.38	"
" " antimony,	41.72	"
" " iron,	2 06	"
Alumina,	4.05	"
Silica,	1.84	"
Silver,	.041	"
	100.091	"

This has the formula $(PbO)_3 (Sb_2O_5)_2 + 4 H_2O$. Although bindheimite, yet it differs from the analysis previously given. Only a few ounces of this were found.

The shaft was sunk to the depth of 32 feet, and at about 18 feet two drifts were started east and west on the vein, but were continued only a few feet. Some excellent ore was taken from these drifts.

Little or no difficulty was experienced in sinking this shaft, as the formation was a black shale, easily removed with the pick, with an occasional blast.

At the bottom of this shaft the vein of quartz which carried the mineral had assumed a thickness of one foot, the ore however occupying only 4 inches. This mine, like nearly every other one in Arkansas, has been worked for only a short season, due, I presume, in most cases to the fact that the parties interested had little or no means to invest in such precarious enterprises, at least in a new mining section with undeveloped and untried veins.

In this instance mining operations were stopped July, 1877, but previous to this time there was a general cleaning up, and the ore was selected and placed in casks for shipment. A large amount was necessarily lost, as this separation was effected by hand. On June 20th and July 5th, 45 casks and 10 casks respectively were gotten ready for shipment, and in all about 25 tons (2240 pounds) were shipped to Messrs. Hallet & Co., London, for which about sixty dollars per ton was paid.

This mine was sold to parties in Memphis in March, 1878, and in the summer succeeding operations were again commenced, but I am not sufficiently informed to report the present prospects or condition of this mine.

Missouri School of Mines, Rolla, Mo., May 2d, 1879.

REGENERATIVE STOVES—A SKETCH OF THEIR HISTORY AND NOTES ON THEIR USE.

BY JOHN M. HARTMAN, PHILADELPHIA.

ON May 19th, 1857, an English patent was granted to E. A. Cowper for heating air or other gases under pressure by means of a regenerator inclosed in an air-tight iron case, having between the regenerator and case a lining of brick. This patent provided for heating the stoves by a separate fireplace, or by gas direct from the blast furnace. A number of forms of interior arrangement of the brickwork are shown in the drawings; also hollow poppet valves with hollow stems, and a pipe inside of the stem for circulating the water; the valve seats have coil cast in them for water circulation to keep them cool; slide valves, with snake coil cast in the discs, are shown, and the use of cold air for cooling the valves is also described. The combustion-chamber of these stoves was central, and openings were provided at the top and bottom to get into the stoves. These Cowper stoves are all circular in section.

November 10th, 1865, an English patent was granted to Thomas Whitwell for regenerative stoves for heating air or gas, provided with cleaning openings at the top and bottom capable of being closed with firebrick plugs and doors. The drawings show a rectangular stove inclosed in an iron case. The interior brickwork has numerous up-and-down passages through the stove, but there is no claim on the interior construction.

March 3d, 1868, an English patent was granted to Charles Cochrane for a slide valve to be subjected to high heats. The disc of this valve was hollow, and had a circulation of water through it by the two hollow stems that operated it. The valve-seat was detachable and had a coil cast in it, through which water circulated. The valve and seat were placed on an incline to the body to cause the valve-disc to lie on the valve-seat. A cap was placed on the bottom of the body to get at the interior readily.

January 5th, 1870, an English patent was granted to Siemens, Cowper and Cochrane for the construction of regenerators in firebrick stoves, with numerous vertical passages of sufficient size to allow a brush to pass through and clean them. These passages had slight projections on the sides to turn the air over and over as it passed through. A claim also covered the use of horizontal passages connected at each end alternately, and the use of blasts or jets of air,

or steam to clean the stoves. This patent was taken out in this country.

July 8th, 1871, an English patent was granted to Thomas Whitwell for a cup under the poppet valves of regenerative stoves to catch the mud deposited in the valve by the water, and keep it away from the valve-face. This patent was taken out in this country.

March 23d, 1872, an English patent was granted to E. A. Cowper for arranging the regenerators of firebrick stoves, whereby the flame passed up and down through the regenerators a number of times. The area of the first passage is large, and that of the subsequent passages smaller, the surface being increased by placing more openings of the same size in the passage. The larger area permits more complete combustion, and the smaller areas provide increased surface to take up the heat. By this arrangement the gas or air passed in the same direction along two or more adjacent walls or partitions. This patent is now being taken out in this country.

August 27th, 1872, an English patent was granted to Thomas Whitwell for upright regenerator walls stayed by cross walls and with cleaning doors on the top and underneath the stove. The air for the combustion of the gas was also heated by passing it through the hollow walls of the regenerator. This patent was taken out in this country.

May 8th, 1874, an English patent was granted to Cochrane and Cowper for the construction of a cylindrical regenerative stove, with an ascending circular flue or combustion-chamber near to one side of the interior of the stove, in combination with a regenerator occupying the remainder of the interior of the stove. The flue and regenerator are so placed that the distances traversed by the air or gas are equal, or nearly so. The apertures of the regenerator passages at the top are narrowed to equalize the distribution of the air or gas. This patent was taken out in this country.

May 16th, 1876, an English patent was granted to Thomas Whitwell for regenerative stoves, with walls or partitions so arranged as to divide the current of air and cause it to pass in the same direction along two or more adjacent walls or partitions. Also for the use of cast-iron pipe on the chimney side of the stove to take up the heat lost at the chimney. This patent was taken out in this country, but the cast-iron pipe is omitted in the American patent.

October 2d, 1877, an American patent was granted to Thomas Whitwell for a water-cooled slide valve, with a detachable valve-seat having a coil cast in it. The valve-disc has also a coil cast in

it, and the valve-face is placed at an angle to the body to cause the valve to lie on the face.

The Siemens-Cowper-Cochrane stove is the result of the combined learning, inventive talent, and practical knowledge of the three men whose names it bears. Dr. Siemens, the physicist, the originator of the regenerative system of heating, is well known to you all; Mr. Cowper is a fertile inventor in many fields, his last notable invention being the transmission of handwriting by telegraphy; Mr. Cochrane comes of a race of ironmasters, and is well known by his able articles on the iron manufacture in English technical journals.

The patents recently granted and now pending in connection with the Siemens-Cowper-Cochrane stoves are improvements in slide valves and the use of compound nozzles to decrease the number of attachments to the stoves; the use of interlocking regenerative brick, and the utilization of the waste tuyere water to wash the gas; improvements in gas washing and the use of overhead flues, with cleaning-doors; the use of piston-surging valves for cleaning the stoves, and improvements in the pipe conveying the hot blast to the furnace; and finally the use of an equilibrium valve worked by a clock attachment to equalize the temperature of the blast during a blow.

The first Cowper stoves could not be cleaned on account of the brick of the regenerator being laid with interstices between them, but with no continuous passage from top to bottom. The stoves worked well when new. The next step was to keep the dust out of the stoves by using large settling chambers containing shelves for catching the gas dust. This helped the stove, but the chambers made additional expense, and they have since been abandoned.

The next improvement consisted in making numerous vertical passages with thin walls in the regenerator, which could be cleaned with a brush, or by jets of air or steam. Still later the vertical combustion-chamber was placed on one side of the regenerator, causing the gas and air to travel the same distance in the stove. The diameter of the stove was diminished and the height increased, which cheapened the stove and gave a better distribution of gas or air over the whole surface of the regenerator.

Mr. Whitwell bought of Mr. Cowper the right to build regenerators inclosed in air-tight casings, and being convinced that his rectangular stoves would bulge out, altered them to the circular (Cowper) form. He also bought of Dr. Siemens the right to use the brick for regenerative purposes, and brought his stoves into practical operation.

Finding the escaping gas in the chimney too high in temperature, Mr. Whitwell placed a number of large iron plates in the back courses of his stoves for the purpose of taking up the waste heat, but after trial they were abandoned. He next placed a series of cast-iron pipes in the back courses of the stove. This involved a separate set of pipes and valves and was found too cumbersome. Mr. Whitwell found it impossible to collect this heat without greatly increasing the size of his stove.

It is here that the advantages of the regenerator with thin walls and large surface is shown, since the lower the temperature the greater must be the surface to take up the heat.

The Siemens-Cowper-Cochrane stove consists of an air-tight iron shell lined with red brick and firebrick, containing a flame-flue near one side, which is partly surrounded by a regenerator. (See Plate I.) The gas is burned in the flame-flue or combustion-chamber, and the products of combustion are then spread over the surface of the regenerator, which absorbs the heat, the escaping gas going off at a temperature of 250° F. when running blast heated to 1200°. The regenerator is formed of brick three inches thick, so arranged as to leave numerous holes four inches square from top to bottom, giving an enormous heating surface with a minimum of brick.

In experiments made with nine-inch walls, the stove was put on a blow for three times in succession without heating it up. After waiting four hours each time, the heat in the interior came out to the surface of the walls, showing that there was a large heat storage that was not available in the two hours the stove was on a blow. If this amount of material had been placed in three-inch walls the stove would have had three times as great capacity in the two hours. This enlarged surface would give a more uniform heat and be more economical of gas.

If we take a No. 1 firebrick, heat it white hot, and drop it into a cistern of water, it will be found that when it has cooled sufficiently to be taken in the hand it is still white hot half an inch from the surface. This shows the slowness with which firebrick yields up its heat, and as three-inch walls during a blow of two hours have proved as efficient as nine-inch walls, heavy walls are evidently useless. A serious defect in firebrick stoves is the glazing of the walls from overheating when the stoves are too small. This glazing prevents heat from being absorbed or given out. This glazing can be overcome by increasing the surface so that the stoves can be worked

at a lower temperature and yet keep the required heat of blast. It is a choice between a mass of thick brickwork at a high heat or a large surface at a moderate heat. The first will glaze the walls and give an irregular heat, the second will not glaze and will give a more uniform heat.

In calculating the heating surface of firebrick stoves only those parts of the walls that do not come into immediate contact with burning gas should be taken as heating surface. Any brickwork exposed to burning gas is invariably glazed when there is perfect combustion, and where gas is burned through the whole length of the stove the efficiency of the stoves is much decreased. The stoves at Crown Point have been subjected to extremely high heats, and after fifteen months running the combustion-chamber was found slightly glazed, just enough to protect the brick, while the regenerator was not at all glazed, the bricks being in the same condition as when first put in. This superficial glazing of the firebrick in the combustion-chamber is an advantage, for the transmission of the heat is thus hindered and the total destruction of the bricks prevented. By confining the combustion to one large chamber, as is done in these stoves, and allowing the gas to expand, the temperature of the gas is lowered, while the volume is increased and glazing in the regenerators prevented. This lower temperature necessitates more surface to take up the heat, which is provided for in the thin walls of these stoves. The capacity of the stoves is measured by the surface of the regenerator alone. Five square feet of regenerator surface is used for each cubic foot of air per minute delivered by the blowing engine. The stoves are so constructed that either the dome, or the regenerator, or the combustion-chamber can be taken out and repaired without interfering with the others, as the heating surface is independent of the roof or side walls.

The old objection to cleaning these stoves no longer exists. With the straight four-inch passages a brush is guided on all four sides, and can be passed through them more rapidly and effectively than in an opening where it is guided on two sides only. Each passage of the brush cleans a space sixteen inches wide, and it takes about thirty seconds to clean an opening. There is one large cap at the top to remove to get at all the openings.

As an ounce of prevention is worth the pound of cure, these stoves are fitted with a plain simple gas-washer which removes nearly all the dust in the gas. There is a small portion of a white flocculent ash that passes the washer, but it is infusible at the highest heat ap-

plied to it, and is easily blown off the walls. The worst enemy of the firebrick stove is the fine ore dust that forms an easily fusible silicate of iron which glazes the walls. The gas-washer catches this dust and keeps it out of the stove as well as the coal tar when raw coal is used. The coal tar condenses on the valve-faces and prevents the valves from opening or from shutting tight. The gas-washer is shown on the elevation (Fig. 1, Plate I); above the washer is a dust-catcher, which relieves the washer of part of its work.

When a furnace works irregularly and slips, large volumes of dust are thrown over, and at such times the dust-catcher is especially valuable. After the gas leaves the dust-catcher it descends and meets the water spray formed by the waste water of the tuyeres falling in three-quarter inch streams on splash-plates, breaking up the water into spray, which falling down through the gas cools it, and thus tends to draw the gas from the tunnel-head. The gas receives a further washing by passing horizontally under the broad flange, after which it slowly ascends through the large annular space, where any entangled moisture may be deposited.

Trial was made of dipping the flange below the water, which washed the gas better, but it threw too much back pressure at the tunnel-head. To wash gas properly a large volume of water is necessary. The more water used the better the washing is done, and the water, moreover, is prevented from getting hot enough to form vapor, which would impair the calorific power of the gas. The water used in the washer should escape with not over thirty degrees additional heat.

Care must be taken in igniting the washed gas. If the valve is thrown open suddenly an explosion is sure to follow. The valve must be opened slowly until the first blue point of flame is seen, after which there is no danger. To ignite the gas a large oil lamp is used with a hollow handle, which keeps the flame supplied with air and prevents the gas putting it out. After passing the washer the gas possesses a higher calorific power, being freed from the dust, which interferes with perfect combustion. Another advantage of washing the gas is found in the fact that dust is not blown into the furnace, where the silica might be reduced if not absorbed by a strongly basic slag.

After a fifteen-months run at Crown Point the furnace with firebrick stoves went out of blast with its hearth cut through. A careful examination of the stoves showed there was no dust in the regen-

erator, each opening being as clean as when started, and there was no dust in the bottom of the stove. This proves that these stoves can be cleaned while running, and that there is no occasion for a ten hours' stop to clean a stove. These stoves had not been entered for cleaning since first starting them.

To burn gas thoroughly combustion must take place in a large chamber with thick walls maintained at a high heat. In these stoves the gas is cut up into strips by passing it through slots in the bottom of the stoves, then mixed with air, and by the time it has travelled forty-five feet to the top of the regenerator it is thoroughly burned. The regenerator is supported on cast-iron gratings, with a space below for draught passage to the chimney and blast passage when the stove is on blast. Ample provision is made in the walls for expansion, and by using an air space between the shell and wall the heat lost through the shell is reduced to a minimum.

Close by the chimney nozzle is a piston blow-off valve to relieve the pressure in the stove when it is to be put on gas. The valve is operated by the blast, so that when the pressure is taken off the piston, the internal pressure in the stove pushes the valve open quickly and lets the blast escape rapidly, which carries out with it any dust that may have fallen to the bottom of the stove as well as the greater part of the dust deposited on the walls when the stove was on gas. As this is repeated every six hours it removes a large amount of dust and helps to maintain the cleanliness and efficiency of the stoves. The cold-blast valve is a pivot valve that can be thrown open or shut instantly. Twice a week, at casting time, the air valve and cleaning valve are opened and the engineer is told to run the engine lively and watch her. When pressure is up the cold-blast valve is suddenly thrown open, allowing the blast to rush through the stove and sweep the dust deposit from the walls. So strong is this current that care has to be taken lest the top course of the regenerator be blown off. To resume, the use of the gas-washer and the piston blow-off valve, and the practice of blowing through the stoves twice a week, prevents accumulation of dust and insures regular working. The patents on these stoves give them the exclusive right of cleaning by this method.

The valves are all plain simple slide valves (see Fig. 5, Plate I), and are modifications of the original Cochrane valve of 1868. The gas and hot-blast valves are water-cooled. The chimney valve is cooled with a current of air drawn through it by the chimney. The valves are hung on one side of the centre, which causes

them to lie against the face and clean off any deposit from it. The gland has a lateral motion, which prevents the stem from binding. By slacking up the four bolts holding the cap and winding up the chain, the valve and cap can be removed in five minutes. A half-inch stream of water supplies each stove, and where not over 1200 degrees of blast is used the valves can be run without water.

To operate these stoves they are opened and gas burned in them four hours, when they are closed and the blast blown through them for two hours. Three hours are required to give the stoves time to heat up. The flues connecting these stoves are all overhead, and can be cleaned at casting-time in ten minutes. Owing to the small amount of gas used the flues are small. This is due to pure gas and large heating surface. On first turning a stove on a blow the temperature of the blast is higher than required, and as the blow continues this temperature falls, and by the end of the blow it is too low. To obviate this and maintain a uniform temperature a connection is made from the cold-blast to the hot-blast pipe. In this pipe is placed a valve with a clock attachment, to gradually close the valve during two hours. At the beginning of the blow this valve is wide open and admits a certain volume of cold air, which cools down the hot blast to the proper temperature. This cold air does not rob the stove of heat, but simply equalizes the temperature of the blast during the time the stove is in air. The amount of air heated per minute (in a blow of two hours) to a given temperature and the temperature of the escaping gas are the measures of efficiency of firebrick stoves.

Many attempts and propositions have been made to substitute other material in the regenerator for firebrick, but without success. The capacity of firebrick for heat, and its slow conducting power, render it most advantageous for the storage of heat, and hence for uniformity in radiation.

Firebrick stoves cannot be destroyed, and under ordinary circumstances they should run ten years without repairs, except the refitting of the valve-faces, which may be necessary every three years. Bricks which have been in use for twenty years in regenerators are found to be still good.

An important feature in firebrick stoves is their ability to cope with damp weather and yet keep the furnace steady. Moisture entering the furnace is decomposed into oxygen and hydrogen, absorbing heat with rapidity, and giving a cold cinder and cold iron. If the hydrogen was burned in the furnace we could get these heat

units again; but experience shows that it escapes as hydrogen and can be detected at the tunnel-head by analysis. When damp weather comes on it is necessary to use a little more gas to keep the blast heat uniform. As the specific heat of the blast is increased by the moisture, it will absorb more heat units from the regenerator. This extra heat will give to the hearth the heat that the decomposition of the moisture would absorb. It has been found that 1250 degrees is a good average temperature for safe running when the furnace is properly burdened. This leaves a margin in the stoves for safety, and allows enough carbonic oxide in the gas to keep up steam and hot blast. If the heat in the hearth is lost by change of cinder, leaky tuyeres, or poor coal, the heat of the blast can in a short time be run up to 1700° and the lost heat restored. The intensity of the heat of the hearth is mainly due to the descending stock, which acts as a regenerator, collecting the heat and delivering it in the hearth. Of the total heat of the hearth, between 60 and 70 per cent. is due to the heat collected by the stock.

If a furnace scaffolds and hangs then the stock does not descend, and no heat arrives at the hearth from this source. It is then that one of the great advantages of firebrick stoves is apparent, for the heat of the blast can be raised to 1700° or 1800°, and the evil quickly corrected. This is the only efficient way of removing a scaffold. With this reserve in the stoves a furnace is always safe, and it should always be maintained. Running in the old style, with plenty of coal, gave a reserve; but for economical running and fast driving the coal must be reduced to a safe minimum.

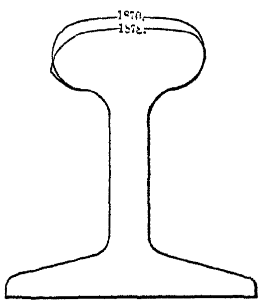
The early advocates of the firebrick stoves supposed that each 100° added to the blast would save at least 200 pounds of coal. Although this has been shown to be an error, yet after careful observation it has been found safe to assume the saving of coal, containing 95 per cent. of carbon, at two hundred weight, between 913°, the limit of the cast-iron stove, and 1250°, the average of the firebrick stove. This saving of two hundred weight is attended with an increase in yield of iron of 12 per cent. without additional cost of labor. With this saving of coal the stoves will pay for themselves in three years and leave a surplus for contingencies by the extra amount of iron made.

The arguments which have been brought forward in opposition to the replacement of iron-pipe by firebrick stoves have, we think, been very effectively answered by the general introduction of the latter in all iron-making countries, where they met with universal favor for safety, efficiency, and economy.

NOTE ON THE WEAR OF AN IRON RAIL.

BY W. E. C. COXE, SUPERINTENDENT OF THE PHILADELPHIA AND READING RAILROAD ROLLING MILL, READING, PA.

At the meeting of the Institute in Philadelphia, in June, 1876, it was my pleasure to read a paper on the "Manufacture and Endurance of Iron Rails." I then spoke of some trial rails which had been placed in the down track of the Philadelphia and Reading Railroad, over which passed, in addition to merchandise and passengers, most of the heavy coal traffic of that company. One of these rails, made at the Philadelphia and Reading Rolling Mill in 1870, and then placed in the track, was removed in 1876, after having carried about 55,000,000 gross tons, and was on exhibition at the Centennial Exhibition. A piece of this worn rail, twisted cold, was also shown there. One of the same series of rails was allowed to remain in the track until 1878, and short sections of this rail, as well as one not worn, to show its original shape, are presented for your inspection. A comparison of the two will show that the top



of the head or tread has worn down $\frac{3}{16}$ths of an inch during its nine years of service. It carried in that time 67,000,000 gross tons of freight, passengers, cars, and engines. This rail was made in the usual way of manufacturing iron rails at the Philadelphia and Reading Rolling Mill. The head was made from puddled iron bars, piled, heated, and rolled into bars, $4\frac{1}{2}$ and 3 inches wide by 1 inch thick, these bars, breaking joints, were made into a pile in section 9 inches square, heated and rolled into slabs 9 inches wide and 2 inches thick; the balance of the rail-pile was made of $4\frac{1}{2}$ and 3-inch bars, rolled from two-thirds old rails and one-third puddled iron, the

whole forming a pile 9 inches square in section. This was heated and bloomed, and then re-heated before rolling in two-high rolls into the finished rail.

The test-pieces from the head-bar of this rail gave a tensile strength of 63,000 lbs. The borings from the head of the worn-out rail analyzed as follows:

Phosphorus,	.422
Carbon,	.027
Silica,	.392
Sulphur,	.032
Manganese,	.164
Iron,	98.963
	100.000

In making comparisons of the endurance or wearing qualities of iron and steel rails only the best of each kind should be taken, and the difference will not be nearly as great as has generally been assumed.

ON THE CLASSIFICATION OF ORIGINAL ROCKS.

BY THOMAS MACFARLANE, WYANDOTTE, MICHIGAN.

During the last fifteen years the progress made in the science of lithology has been of a very marked character. From being a miscellaneous collection of facts and observations regarding certain mineral aggregates seemingly incapable of systematic arrangement, it has become a most interesting and orderly department of science. The investigators who have brought about this great change have recorded the results of their labors, and placed at our service an extensive and varied literature on the subject. To this I have paid some attention, and partly from studying it, partly from my own observations, I have wrought out a classification of original or crystalline rocks, which I venture to bring under the notice of the members of this Institute. The practical importance of the subject will be acknowledged when it is considered that rock species are not without their influence on the contents of the mineral veins which intersect them. Besides, the Institute has been doing good service in defining terms in metallurgy, and no doubt in mining and geology we might

be able to understand each other better if we could agree as to the signification of the various lithological terms now in use.

My first attempt in this direction was made in 1872, and I then published in the *Canadian Naturalist* a table showing the constitution of the various species of original rocks. This table, very much modified and I think improved, accompanies the present paper, and will serve to show the principal features of my system.

It will at once be observed that there are excluded from the table and from consideration in this paper three other classes of rocks, which may be characterized as follows: *Altered rocks*, produced by the chemical alteration of original rocks in place; *derived rocks*, built up of the ruins of pre-existing rock-masses or strata; and *metamorphic rocks*, resulting from the action of various agents, and especially of heat, on rocks of the last-mentioned class.

The table is, in the main, so constructed as to explain itself, but still its ruling principles will require to be pointed out. The various species of rocks mentioned in it are not supposed to be sharply separated from, but rather to pass gradually into each other. The same thing happens with several series of minerals, and yet we have definite compounds withdrawn, as it were, from the continuous succession, and emphasized as distinct species. The present tendency of lithologists in this matter of classification is to ignore division lines altogether, and to regard rock species as subject to insensible transitions in every possible direction. In proof of this may be mentioned the difficulty which Rosenbusch has recently raised as regards the line of demarcation betwixt granite and felsite, the desire on the part of some investigators to obliterate the distinction between massive rocks of Tertiary and pre-Tertiary age, and the abandonment by Zirkel himself of the separation which he made in his classical *Lehrbuch* between anorthitic greenstones and those carrying the more siliceous triclinic felspars. That transitions do occur in the texture as well as in the chemical composition and mineralogical constitution of rocks is of course apparent to every one; but that there are well-defined types which stand out in clear relief from each monotonous transition series I am very much disposed to maintain. The rock species named in the body of the table are I believe such types or nuclei, each surrounded by a penumbra of less decided varieties passing into the adjoining species, as the texture, components, or minerals gradually change.

From the table it will be seen that eleven different modifications of *texture* have been distinguished, and that this characteristic ranks

first in my scheme as a means of separating into different orders the rocks now under discussion. Here I may remark that it seems in the highest degree reasonable that this character should first have consideration. It is the first distinction we naturally make on looking at a rock specimen, and the oldest names in lithology—granite, micaschist, porphyry, trachyte, etc., have reference only to this feature. We prefer, with Naumann, Cotta, and Zirkel, the term "texture" to "structure," because it has reference to the manner in which the rock constituents, which are often minute fibres or laminæ, have been, as it were, woven together. "Structure," on the other hand, seems a convenient term for indicating the peculiar way in which the larger masses of a rock are built together, as, for instance, in the case of the columns of basalt or the globular concretions of many porphyries. It is, I think, to be regretted that Rosenbusch has recently* substituted the word "structure" for the term "texture," which has been used to indicate this character by almost all his predecessors.

To define the terms used in the table for distinguishing the various sorts of texture would scarcely be necessary, were it not that, owing to the indefatigable labors and resulting new views of microscopical lithologists, our ideas regarding texture are in danger of becoming confused. It may therefore be profitable to indicate briefly what I consider to be the limits of each variety of texture.

In schistose rocks (I) the constituents are supposed to be large enough for determination by the naked eye. In slaty rocks (II) they can only, if at all, be determined under the microscope. Similarly, among the large and small grained order (III), the constituents are supposed to be granular, and never to sink to such a size as to require the microscope or even the magnifying glass for their determination. When this, however, takes place with any rock it becomes fine-grained (IV), and although the microscope may unveil its mineralogical constitution it is no longer entitled to be regarded even as small-grained (*Kleinkörnig*). When porphyritic structure is developed in a small-grained rock, porphyritic granites, syenites, or diorites result, which are only to be regarded as passage members between the typical granular and porphyritic orders. When, however, the larger crystals are developed in a fine-grained matrix, the result is assumed to be a true porphyry, and this although the matrix should be capable of complete individualization under the microscope. In thus defining porphyritic texture (V) I adopt the view long ago enunciated by Von Buch, that the matrix of a porphyry is always made up of

* Rosenbusch, Microscopische Physiographie der Massigen Gesteine, 1877.

different minerals, although sometimes exceedingly minute. When the constituents of the matrix decrease in size to such an extent that under the microscope they can no longer be recognized even as a micro-crystalline aggregate, and when a vitreous base of some description can be detected, the resulting rock cannot any longer be termed a porphyry. This is, of course, opposed to the new view of Rosenbusch, who proposes to exclude from the porphyries all the rocks of that order which do not possess a vitreous base. The adoption of this proposal would, it appears to me, introduce the greatest confusion into lithology, and leave the field geologist almost helpless in his work. Rocks which possess an unindividualized base very frequently betray their nature, even macroscopically, by their vitreous appearance. This is the case with tachylyte and pitchstone, which I regard as belonging to the hyaline order of texture (VI). The term vitreous might also be used, but it would seem best to reserve that name for thoroughly anhydrous rocks, and to associate the idea of hydration with the word hyaline. Amygdaloidal (VII) and spherulitic (VIII) texture do not require to be described here. Their general peculiarities are well known, although it might be difficult to indicate exactly the dividing line between them. It is also a question as to how far spherulitic and variolitic texture are of the same nature. As regards trachytic (IX) rocks it may be remarked that when Haüy originally introduced the term trachyte he had reference more to a peculiarity of texture than of mineralogical constitution. Deville's proposal to return to this definition and use the word chiefly to indicate texture seems extremely reasonable. In this order all rocks are included, with the rough fracture, porous matrix, and glassy or fissured crystals which trachytes usually possess. They are distinguished from the porphyries by the presence of an unindividualized base in greater or less quantity. When the unindividualized base or glass of trachytic rocks very much increases in quantity the result is a purely vitreous texture (X). On the other hand, when their character of porosity is intensified and scoriaceous rocks are produced, it would seem appropriate to indicate this as cavernous texture (XI).

Next in importance to texture in this classification of original rocks stands chemical composition. It is certainly not such a palpable feature as texture, unless, indeed, color is regarded as to some extent its index. The light-colored, *i. e.*, white, gray, or reddish rocks, are almost invariably more siliceous than those of a green or black color; but still it would scarcely be practicable to make use of this charac-

ter in classification. Heretofore, mineralogical constitution has been the ruling principle in determining rocks, but there are several considerations which would seem to justify its subordination to chemical composition. The most important of these is the very generally accepted idea among lithologists that original rocks have resulted from the cooling and solidification of certain fused glasses or slags, composed of silicates of the alkalies, alkaline earths, and certain metallic oxides, and that the chemical composition of these fused masses mainly determined the mineralogical nature of the resulting rock. Further, although the constituent minerals have been very advantageously used in characterizing species, they have not proved so useful in distinguishing the various rock families, and we propose to show that chemical composition is much better fitted for this purpose. Bunsen was probably the first to bring this characteristic into notice, and afterwards, as the number of rock analyses increased, it has come more and more into prominence. Just as we distinguish between essential and auxiliary constituents in rocks, we can also in a similar manner separate the essential and auxiliary components. The former I conceive to be silica, alumina, ferrous and ferric oxides, magnesia, lime, soda, and potash, while the rarer substances, such as zirconia, oxides of manganese, boracic acid, titanic acid, etc., may be regarded as non-essential.

On reference to the table, it will be found that we have distinguished the silicic, siliceous, neutral, basous, and basic rock families, which terms are, of course, of chemical origin. In indicating their composition more closely it does not seem advisable to make use of the ordinary chemical nomenclature, which, for a neutral or monosilicate, requires the presence of an atom of silica to each atom of protoxide base. The system employed by metallurgists in describing their slags, and by mineralogists to indicate the composition of mineral species, appears also to be well fitted for application in lithology. According to this system the various rock families would be constituted as follows:

Names of Family,		Oxygen Ratio.		
Chemical.	Lithological.	Bases.		Acid.
Silicic, . . .	Persilicates, ...	1	:	4 and over.
Siliceous, . .	Tersilicates, ...	1	:	3
Neutral, . .	Bisilicates, ...	1	:	2
Basous, . . .	Sesquisilicates, . . .	1	:	1½
Basic, . . .	Subsilicates, ...	1	:	1¼ and under.

By searching among the analyses of granular rocks now at our disposal for those species whose oxygen ratio corresponds most

closely to those just mentioned, we find their localities, oxygen ratios, and silica contents to be as follows:

Name of Family.	Name of Species.	Locality.	Oxygen Ratio Bases.	Silica.	Percentage Silica.
Persilicates, . .	Granite, . .	Blackstairs Mountain, . .	1	: 4.100	73.20
Tersilicates, . .	Grantyte, . .	Meineckenberg, .	1	: 2.907	66.81
Bisilicates, . .	Syenite, . .	Bergstrasse, . .	1	: 2.092	59 80
Sesquisilicates, .	Dioryte, . .	Rosstrappe, . .	1	: 1.544	51.07
Subsilicates, . .	Eukryte, . .	Carlingford, . .	1	: 1.272	47.52

Most of these and the following analyses are taken from Zirkel's *Petrographie*, but King's works on the Fortieth Parallel Survey have also been consulted.

In a similar manner the porphyritic rocks yield the following series:

Name of Family.	Name of Species.	Locality.	Oxygen Ratio Bases.	Silica.	Percentage Silica.
Persilicates, . .	Felsitophyre, . .	Sandfelsen, . .	1	: 3.934	70.85
Tersilicates, . .	Porphyrite, . .	Wilsdruff, . .	1	: 2.988	67.25
Bisilicates, . . .	Rhombporphyry, .	Vettakollen, .	1	: 2.070	56.00
Sesquisilicates, .	Melaphyre, . . .	Bährethal, . . .	. .	. .	54.26
Subsilicates, . .	Augitophyre, . .	Fassathal, . . .	. .	. .	45.05

As regards melaphyre and augitic porphyry the ratios have not been given, because it is impossible to find instances of these rocks which have not undergone alteration.

Analyses of trachytic rocks approaching most closely to the same oxygen ratios give the following results:

Name of Family.	Name of Species.	Locality.	Oxygen Ratio Bases.	Silica.	Percentage Silica.
Persilicates, . .	Rhyolyte, . .	Hohenburg, .	1	: 4.021	72.26
Tersilicates, . .	Trachyte, . .	Wahsatch, . .	1	: 2.950	64.93
Bisilicates, . .	Andesyte, . .	Sandec, . . .	1	: 2.095	58.11
Sesquisilicates, .	Anamesyte, .	Esjaberg, . .	1	: 1.534	50.05
Subsilicates, . .	Basalt, . . .	Kreuzberg, . .	1	: 1.210	44.85

If we take the foregoing instances we obtain the following percentages as the average silica contents of the various rock families:

Persilicates, ...	72.10
Tersilicates, ...	66 33
Bisilicates, ...	57.97
Sesquisilicates, ...	51.79
Subsilicates, ...	45.81

The increase in the silica from the basic to the silicic family in the foregoing list averages 6.57 per cent., and from this it would seem reasonable to confine the variations in each family to 7 per cent., as

has been done in constructing the table. This method of drawing division lines may appear somewhat arbitrary, but it has been found to agree tolerably well with the results of rock analyses.

As has been already mentioned, mineralogical constitution is considered in our system as exclusively applicable to the determination of species. To use this character, as has been frequently done, to separate rocks into groups does not seem to lead to satisfactory results. Roth and Zirkel have grouped the crystalline rocks in accordance with the nature of their felspars, but it is very doubtful whether such a system can be thoroughly applied. Apart from the difficulty of distinguishing the various felspars in rocks, the frequent simultaneous occurrence of more than one of these causes confusion in limiting the various groups. Even in distinguishing species the felspars would seem to be of far less utility than the relatively more basic constituents, such as mica, hornblende, etc., and the latter seem to have been more frequently made use of in former times for this purpose. The minerals which have been admitted into the table are all made up of the chemical components above mentioned, and species containing the rarer acids and bases have been excluded. The following are therefore regarded as the essential constituents of original rocks:

1. Quartz.
2. Orthoclase.
3. Oligoclase.
4. Labradorite.
5. Anorthite.
6. Nephelite.
7. Leucite.
8. Garnet.
9. Epidote.
10. Muscovite.
11. Sericite.
12. Biotite.
13. Amphibole.
14. Pyroxene.
15. Smaragdite.
16. Diallage.
17. Hyperisthene.
18. Enstatite.
19. Crysolite.
20. Hematite.
21. Magnetite.

The first five of these minerals, even although quartz is one of them, may be called the felspathic constituents, and the remaining sixteen, being relatively much less acid, may be called basic constituents. They are thus distinguished in the table, which shows the various rock species resulting from their association. The number of these amounts to one hundred and fifteen, and includes all the original rocks known to lithologists. As far as has seemed advisable Dana's termination *yte* has been used in order to distinguish easily between rock and mineral species. By following each horizontal line in the table, to the left of any of the species named, its basic constituents, texture, and approximate age may be ascertained, while

the reading at the head of the vertical column containing it gives its felspathic constituents and chemical characters. Each horizontal series forms a transition from right to left of silicic to more and more basic rocks, while each vertical series, besides indicating the passages caused by the substitution of one basic constituent for another, shows also the differences in rocks of similar chemical composition caused by change of texture. The place assigned to each of the species in the table is generally in accordance with the views of the highest authorities in lithology. Where these differ the writer has adopted the view which seemed to him most reasonable. In only two cases have new names been introduced, apart from some slight abbreviations of those already existing. The two new names are anorthophyre, to indicate the anorthite porphyry discovered by me near Thunder Cape, Lake Superior, and Raphaelyte, a compound of anorthite and magnetite, to which my attention was called by my friend Mr. Raphael Pumpelly. It is possible that a slightly different signification has been attached to some of the old names in the table, especially in the case of synonyms, and it may be as well to point out these. Corsilyte and euphotide, as well as granitone and gabbro, hyperite and norite, anamesyte and dolerite, at present merely indicate the more and less siliceous varieties of the same rock, but it is quite possible that, by distinguishing the different plagioclases in these rocks, their complete independence as species may yet be established. The first of these remarks regarding the degree of acidity applies to felsyte and euryte, greenstone and trap, pitchstone and retinyte. It will be observed that the termination *phyre* has an extended application as equivalent to porphyry. Zirkel's old definition of melaphyre has been adhered to rather than that of Rosenbusch, and the synonym, basaltyte, has been used to indicate the crysolitic porphyries. Among trachytic rocks the terms liparyte and rhyolyte have heretofore been regarded as quite synonymous. Since, however, the rocks of the Lipari Islands are generally more micaceous than Von Richthofen's rhyolytes, it would seem reasonable to apply the name liparyte to those silicic rocks of the trachytic order which contain mica, and restrict the term rhyolyte to those in which it cannot be detected. Both names are used in our table in a much more restricted sense than their originators intended, for both Roth and Von Richthofen included in liparyte and rhyolyte rocks of a hyaline, spherulitic, and vitreous texture. The name andesyte has been also restricted in our table, and includes the hornblende andesites only. For those containing augite, Abich's old name for these rocks, trachydoleryte is applied.

I do not know that I can profitably add anything further in explanation of the table now submitted to the Institute. That my classification will be found in all respects satisfactory I do not expect, but I hope that it may be found serviceable in some degree until superseded by a better. I am also sanguine enough to believe that some of my fellow-members may, by using it, ascertain at a glance what otherwise might occasion them several hours' study.

WYANDOTTE, MICH., April 12th, 1879.

ON THE USE OF DETERMINING SLAG DENSITIES IN SMELTING.

BY THOMAS MACFARLANE, WYANDOTTE, MICH.

IN smelting copper, lead, and silver ores, it is scarcely possible in every case to make analyses of the various parcels of ore, with the view of combining these and the fluxes so accurately as to yield, in the furnace, slags of exactly the most favorable composition. This is even more difficult with ores which require previous calcination, or with mattes which have been roasted in heaps. Even in cases where the greatest pains have been taken and the most elaborate calculations made beforehand, it frequently happens that variations in the working of the furnace interfere with the result so much desired. Generally, the practical metallurgist must be content with ascertaining the average composition of his ores and fluxes, making a calculation once for all as to the most advantageous mixture, and leaving slight changes in its composition to be taken care of as it is passing through the furnace. The various products then afford him the best material whereon to base his judgment as to the manner in which his smelting mixture is working. Of course the character of the slag is one of his chief guides, but it is not always possible from its outward characters alone to form a correct judgment as to its composition. Neither would it be possible or practicable to apply chemical analysis for this purpose, as it would be impossible to wait for its results while working a furnace.

Abich was, I believe, the first to point out the relation existing between the composition of volcanic rocks and their densities, and to show that, the latter being ascertained, very correct conclusions

might be drawn as to their contents in silica. The general rule is that the most siliceous rocks are the lightest, and the most basic have the highest specific gravity. It occurred to me that by determining the density of slags it might be possible to judge more accurately of their nature than by observing merely their superficial appearance. This plan was first applied at the Wyandotte Silver Smelting and Refining Works in September, 1877, when siliceous silver ores and galenas from Georgetown, Colorado, were being treated. Mr. F. H. Williams determined the following densities and silica contents of the slags from these ores:

Sp. Gr.	Per cent. SiO_2.
3.44	37.60
3.48	36 35
3.50	36.30
3.51	35.80
3.56	35.15
3.57	33.90
3.63	31.15
3.88	30.40

In July and August, 1878, Mr. S. B. Wight made the following examinations of three slags from one and the same smelting campaign, during which Western ores were being treated:

Sp. gr.	Per cent. SiO_2	Al_2O_3.	FeO.	CaO.
3.47	35.05	Undetermined.	31.51	15.77
3.52	32.56	"	37.70	14.68
3.60	31.96	"	36.97	12.71

The gradual increase of density as the silica decreases in both these series of examinations will be apparent. But the rule is not absolute, and is only applicable where the relative quantity of the different bases in a slag remains the same. In smelting ferruginous ores and lead and copper mattes the quantity of protoxide of iron in the slag in proportion to the other bases is usually greater, and this increases the density, although the percentage of silica may remain the same. A slag from smelting lead matte gave, on examination by Mr. S. B. Wight, as follows:

Specific gravity,	3.65
Silica,	34.67
Alumina,	14.85
Ferrous oxide,	39.04
Lime,	7.54

If this example is compared with the first in the last-mentioned

series it will be found that an increase of 7.53 per cent. in the ferrous oxide, while the silica remains nearly the same, causes a considerable increase in density. The following determinations are further generally illustrative of the increase of density in slags as their percentage of silica decreases:

Where produced.	Sp. gr.	SiO_2.	Al_2O_3.	FeO.	CaO.
Iron Blast Furnace, Wyandotte,	2.85	55	Undetermined.		
Copper Works, Detroit,	3.04	43.31	26.22	——	27.40
Germania Works, Utah,	3.81	28.01	Und.	48.10	12 37
Eureka Consolidated Works, Nevada,	4.18	26 47	"	61.62	2.73
Wyandotte Rolling-Mills ("heating slag"),	4.29	25.49	"	75.06	——

Some of these determinations are by Dr. Hermann Hahn, others by Mr. S. B. Wight.

In practically making use of the slag densities for regulating the smelting at Wyandotte Silver Smelting and Refining Works, Jolly's spring balance (described at page 59 of Brush's *Determinative Mineralogy*) was found of great service. Five minutes only are used in making, by its use, two determinations of the density of a slag, and then it is found easy to decide whether any change in the smelting charge is necessary. The smelting of ore goes on regularly and cleanly when the specific gravity of the slag is kept between 3.6 and 3.8. Water-jacket furnaces are used, with six tuyeres, and a height from these to the charging-door of 12 feet. The crucible of each furnace is 18 inches deep, and furnished with a tapping-hole at the bottom, besides the usual lead-well and slag-spout. The tapping occasions no difficulty, and so long as the slag is not allowed to become too heavy no incrustations are deposited in the furnace, which is easily cleaned out at the end of each campaign. The charging is managed in the following way: The smelting mixture is made up of ore and fluxes, and is calculated to yield a slag containing 32 per cent. silica. The slag necessary to keep the charge open is added separately in the proportion of about one-third of the smelting mixture. The slag used is unclean slag from previous campaigns and puddling slag or heating slag (*frischschlacke*) from the rolling mills. When the furnace yields a slag of greater density than 3.8 the proportion of rolling-mill slag is decreased. The latter is on the other hand increased when the slag produced is lighter than 3.6. By ascertaining the specific gravity of the slags frequently, and altering the charge in the manner described, the furnaces are found to work regularly and satisfactorily.

It is not supposed that the exact figures above given will be found

of great value in other smelting works, or that by attending only to the composition of slags in smelting it will be possible to do the best of work. The metallurgist has very many points to consider at one and the same time while conducting smelting operations, and the object of this paper is merely to direct his attention to one of these points and to indicate a mode of ascertaining approximatively the amount of silica in slags with comparative ease and accuracy.

WYANDOTTE, MICH., April 21st, 1879.

PHOSPHORUS IN BITUMINOUS COAL AND COKE.

BY ANDREW S. MCCREATH, HARRISBURG, PA.

THE manufacture of pig iron for conversion into steel by the Bessemer and open-hearth processes, is now one of the most important industries of the United States. It is necessary that iron intended for this purpose should be very pure, and especially must it be comparatively free from phosphorus. Great care must therefore be exercised in the selection of proper ores, flux, and fuel. Only such ores as are practically free from phosphorus can be used, and pure fuel is as much a necessity as pure ores; though hitherto iron men have paid comparatively little attention to this point.

During the course of my work as chemist for the Second Geological Survey of Pennsylvania, I had occasion to examine some of the bituminous coals of the State for phosphorus, and the results obtained are so interesting that I venture to present them to the notice of the members of the Institute.

The coals are arranged in geological order according to the different beds, and the table shows the percentage of phosphorus in the coal and also in the coke.

The greatest number of specimens have been selected from the Pittsburgh bed, because it is the principal coal-bed of Southwestern Pennsylvania, and most of the mineral fuel which is mined along the Youghiogheny and Monongahela rivers, to be used in the coke ovens of the Connellsville region and in the blast furnaces and mills of Pittsburgh and its vicinity, and to be shipped to Western and Southern markets, comes from this bed.

It will be noticed that many of the specimens examined contain

phosphorus in most objectionable quantities. In the twenty-four coals tested from this bed, the amount of phosphorus varies from a mere trace to .1248 per cent., equal to .2003 per cent in the coke. Such a coke could not, of course, be used in the manufacture of Bessemer pig iron, and it is believed that in many cases unsatisfactory results have been obtained simply by the use of an impure fuel.

Table Showing the Percentage of Phosphorus in Certain Coals.

	Name of Coal.	County.	Coal bed.	Phos. per ct. in *coal.*	Phos. per ct. in *coke.*
1	Henderson's, Buffalo Township	Washington.	Washington.	.1667	.2818
2	Lucas's, Dunkard Township	Greene.	Sewickley.	.0053	.0084
3	Miller's, Dunkard Township	"	Pittsburgh.	.0025	.0041
4	Magee's, Independence	Washington.	"	.0254	.0438
5	Ashurst's, Chartiers Township	"	"	.0491	.0846
6	Redd's, Fallowfield Township	"	"	.0943	.1551
7	New Eagle Works, Carroll Township	"	"	.0013	.0020
8	White's, East Pike Run Township	"	"	.1248	.2003
9	Slocum's, East Pike Run Township	"	"	.0011	.0018
10	Penn Gas Coal Co.'s Youghiogheny Shaft	Westmoreland.	"	.0058	.0095
11	Penn Gas Coal Co.'s Penn Shaft	"	"	trace.	trace.
12	Penn Gas Coal Co.'s Sewickley Shaft	"	"	trace.	trace.
13	Westmoreland Coal Co.'s Southside Mine	"	"	.0092	.0150
14	Westmoreland Coal Co.'s Larimer Mine	"	"	trace.	trace.
15	Westmoreland Coal Co.'s Foster Mine	"	"	.0402	.0652
16	Millwood Coal Co.'s, Derry Township	"	"	.0801	.1177
17	Saltzburg Coal Co.'s, Loyalhanna Township	"	"	.0307	.0452
18	Saxman & Co.'s, Derry Township	"	"	.0167	.0247
19	Greensburg Coal Co.'s, Hempfield Township	"	"	.0070	.0107
20	Frick & Co.'s, Connellsville Township	Fayette.	"	.0111	.0161
21	Townsend's, Perry Township	"	"	.0022	.0034
22	McCormack Heirs, Franklin Township	"	"	trace.	trace.
23	Swan Heirs, North Union Township	"	"	trace.	trace.
24	Kendal's, German Township	"	"	.0020	.0031
25	Saylor Hill, Summit Township	Somerset.	"	.0058	.0074
26	Wilhelm Mine, Elk Lick Township	"	"	.0122	.0156
27	Coleman Brothers, Valley Township	"	Berlin.	.0105	.0135
28	Cotter's, Raccoon Township	Beaver.	Bed E.	.0058	.0094
29	Dysart & Co.'s, Washington Township	Cambria.	"	.0530	.0688
30	Dennison, Porter & Co.'s, Allegheny Twnsp	Blair.	"	.0075	.0103
31	Diehl's, Green Township	Beaver.	Bed D.	trace.	trace.
32	R. J. Hughes & Co.'s, Decatur Township	Clearfield.	"	.0080	.0107
33	Rockhill Iron & Coal Co.'s, Carbon Twnsp.	Huntingdon.	"	trace.	trace.
34	Joseph Ramsey, Jr.'s, White Township	Cambria.	Bed C.	.0078	.0105
35	Dennison, Porter & Co.'s, Allegheny Twnsp	Blair.	Bed B.	.0053	.0072
36	Cambria Iron Co.'s, Allegheny Township	"	"	trace.	trace.
37	Cambria Iron Co.'s, Conemaugh Township.	Cambria.	"	trace.	trace.
38	Dysart & Co.'s, Washington Township	"	"	trace.	trace.
39	Brotherline's, Clearfield Township	"	"	trace.	trace.
40	Savage Colliery, Todd Township	Huntingdon.	"	.0080	.0098
41	Connellsville Coke, Frick & Co	"	Pittsburgh.	....	.0140
42	Connellsville Coke, J. F. Dravo	"	"	....	.0140
43	Connellsville Coke, J. F. Dravo	"	"	....	.0130

ON AN APPARATUS FOR TESTING THE RESISTANCE OF METALS TO REPEATED SHOCKS.

BY WILLIAM KENT, M.E., PITTSBURGH, PA.

MORE than twelve years were spent by Wöhler at the instance of the Prussian Government in experimenting upon the resistance of iron and steel to repeated stresses. The results of his experiments are expressed in what is known as Wöhler's law, which is given in the following words in Du Bois's translation of Weyrauch:*

"Rupture may be caused not only by a steady load which exceeds the carrying strength, but also by repeated applications of stresses, none of which are equal to the carrying strength. The differences of these stresses are measures of the disturbance of continuity, in so far as by their increase the minimum stress which is still necessary for rupture diminishes."

A practical illustration of the meaning of the first portion of this law may be given thus: If 50,000 pounds once applied will just break a bar of iron or steel, a stress very much less than 50,000 pounds will break it if sufficiently often repeated.

This is fully confirmed by the experiments of Fairbairn and Spangenberg, as well as by those of Wöhler; and, as is remarked by Weyrauch, it may be considered as a long-known result of common experience. It partially accounts for what Mr. Holley has called the "intrinsically ridiculous factor of safety of six."

Another "long-known result of experience"—although this has not, as the writer believes, been investigated by scientific experimenters—is the fact that rupture may be caused by a succession of *shocks or impacts*, none of which alone would be sufficient to cause it. Wrought iron will crystallize by repeated blows in service and become weaker than cast iron. Iron axles, the piston-rods of steam-hammers, and other pieces of metal subject to continuously repeated shocks invariably break after a certain length of service. They have a "life" which is limited. Iron rods in bridges sometimes crystallize and break, although the rods in most of our iron bridges

* Strength and Determination of Dimensions of Structures. By Dr. J. J. Weyrauch. Translated by Du Bois, New York, 1877.

of to-day may have five, fifty, or five hundred years of life yet allotted to them, due to their factors of safety.

Several years ago Fairbairn wrote: "We know that in some cases wrought iron subjected to continuous vibration assumes a crystalline structure and that the cohesive powers are much deteriorated, but we are ignorant of the causes of this change." We are still ignorant, not only of the causes of this change, but of the conditions under which it takes place. Who knows whether wrought iron subjected to very slight continuous vibration will endure forever? or whether to insure final rupture each of the continuous small shocks must amount at least to a certain percentage of the single heavy shock (both measured in foot pounds), which would cause rupture with one application? Wöhler found in testing iron by repeated stresses (not impacts) that in one case 400,000 applications of a stress of 500 centners to the square inch caused rupture, while a similar bar remained sound after 48,000,000 applications of a stress of 300 centners to the square inch.

Who knows whether or not a similar law holds true in regard to repeated shocks? Suppose that a bar of iron would break under a single impact of 1000 foot pounds, how many times would it be likely to bear the repetition of 100 foot pounds, or would it be safe to allow it to remain for fifty years subjected to a continual succession of blows of even 10 foot pounds each?

These queries are not merely fanciful speculations, they are questions of immense practical importance to engineers, the more so since the latter are beginning to use steel in situations in which iron alone has hitherto been used. You build a steel bridge, the tension of members of which you specify shall have 60,000 pounds tensile strength and 25 per cent. elongation. Are you sure that such a steel will have a longer life than one having 75,000 pounds tensile strength and 15 per cent. elongation, by which you might reduce the weights 25 per cent. without reducing the factor of safety under steady load?

You put steel axles under a car, and specify that they shall bend and not break under five blows of a drop-weight falling a distance of twenty feet. Do you know whether or not an axle which would break at the second blow might not have twice as long a life in service as the one which did not break with five blows? I venture to say that in the present state of our knowledge no one can answer these questions in either the affirmative or negative. No engineer dares to use steel of 100,000 pounds tensile strength in the tension

members of a bridge, not because he *knows* the small elongation accompanying that strength would render the bridge unsafe, but because he *fears* it.

The only actual experiments that have been made to determine the relative resistance to repeated shocks of steel of different degrees of hardness, as far as the writer is aware, are those made by a member of this Institute, Mr. William Metcalf, the results of which are published in the *Metallurgical Review* for December, 1877 (vol. i, p. 399). It is surprising that these results have not yet been made the subject of discussion among engineers. In view of the opinion generally held that soft steel only should be used to resist repeated shocks, they are important and even startling. One set of his tests may here be briefly mentioned.

Some small steel pitmans were made, the specifications for which required that the unloaded machine should run four hours and a half at the rate of 1200 revolutions per minute before the pitmans broke. The steel was all of uniform quality except as to carbon. Here are the results:

The .30	carbon ran	1	hour and	21	minutes,	heated and bent before breaking.
.49	"	1	"	28	"	" " "
.53	"	4	hours	57	minutes,	broke without heating.
.65	"	3	"	50	"	broke at weld, where imperfect.
.80	"	5	"	40	"	
.84	"	18	"			

·87 carbon broke in weld near the end.

.96 carbon ran 4 hours and 55 minutes and the machine broke down.

Some other experiments by Mr. Metcalf confirmed his conclusion, viz.: that high carbon steel was better adapted to resist repeated shocks and vibrations than low carbon steel.

These results, however, would scarcely be sufficient to induce any engineer to use .84 carbon steel in a car axle or a bridge rod. Further experiments are needed to confirm or overthrow them. A scientific investigation of the subject should be made, similar to that made by Wöhler, in regard to resistance to repeated steady stresses. A small fraction of the time and money spent by Wöhler would probably settle for all time to come such questions as these: "Shall we use high or low carbon steel for structural purposes?"

With these suggestions I now present in conclusion an apparatus designed to test directly the relative resistance of metals to repeated

shocks. I believe it will increase our knowledge concerning some of the following questions:

1. Is steel better adapted than wrought iron to resist repeated shocks?

2. Does steel crystallize and become weak under repeated shocks as does wrought iron?

3. Is high or low carbon steel best adapted to resist repeated shocks?

4. What is the influence of other chemical constituents than carbon upon this resistance?

5. What relation exists between the strength and ductility of metal, as determined by the ordinary tensile testing machines, and its resistance to repeated shocks?

6. What relation exists between resistance to single heavy shocks (as given by the drop test) to the resistance to repeated shocks?

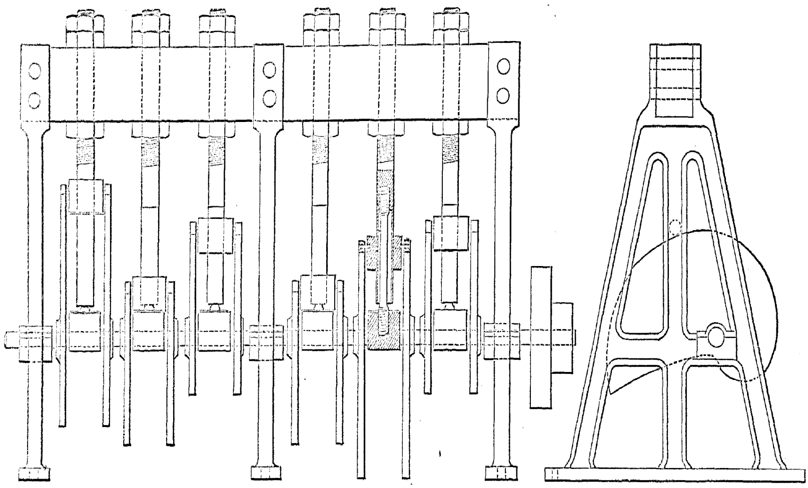
7. Is the same factor of safety necessary in steel as in wrought iron?

The apparatus is a machine by which metals may be tested by repeated blows, numbering millions if necessary.

The test piece is a round rod of convenient length and diameter, and the blows are applied in the direction of the length, as in tension tests. A small nut or anvil is screwed or otherwise fastened on the bottom of the rod, and an annular weight is caused to drop repeatedly upon the anvil by means of a rotating cam. The whole machine is built as rigidly as possible to avoid spring, and the anvil being made small to avoid absorption by its inertia of the energy of the blows, the whole of the energy of each blow is transmitted to the test piece, straining it in the direction of its length. The weight of the drop being known and the height of fall being uniformly adjusted, each blow gives an impact of the same number of foot pounds. A revolution counter attached to the driving-shaft indicates the number of blows that are given.

In the particular machine shown in the drawing six tests may be made at once. Each test piece is supported by being screwed or otherwise fastened into a holder or sleeve, the upper end of which is screwed and held with jam nuts in the heavy overhead beam. A revolving shaft carrying six double cams raises and lets fall an annular weight upon the anvil attached to each test rod. Sixty blows per minute on each rod may easily be given, making 3600 in an hour, 86,400 in a day, and more than half a million in a week. The cams are placed at different angles upon the shaft so as to avoid inducing

large vibration in the framework. A brass tube fitted to the bottom of each sleeve serves as a guide for the annular weight.



In the drawing the test pieces are ten inches in length between shoulders and one inch in diameter, the drop weighs from fifteen to twenty-five pounds, and the fall is ten inches. The diameter of the test piece and the weight of the drop should be adjusted so as to allow the piece to break at a convenient number of blows, and after being once adjusted they should be kept uniform, in order to obtain the relative resistances of different kinds of metal. The fall can be kept nearly uniform, notwithstanding the elongation of the piece during test, by adjusting the nuts on the sleeves.

The power required to drive the machine, testing six pieces at once, 60 blows each per minute, the drop weighing twenty-five pounds, with fall about ten inches, is about one-fourth of a horse-power. The convenience of operation is all that can be desired. When a piece breaks, the lower portion with the anvil falls to the ground without interfering with other tests.

All the attendance needed is that sufficient to make a record of the figures on the revolution counter, to take up elongation by screwing up the nuts on the sleeves, and to replace broken test pieces.

The complete machine stands four feet high, weighs about 1800 pounds, and will cost from $200 to $300.

ON SOME CURIOUS PHENOMENA OBSERVED IN MAKING A TEST OF A PIECE OF BESSEMER STEEL.

BY WILLIAM KENT, M.E., PITTSBURGH, PA.

ABOUT a year ago, the writer had occasion to assist Mr. John L. Gill, Jr., of the Pittsburgh Car-wheel Works, in making a trial of his new testing machine. A piece of Bessemer steel, of about .34 carbon, was used as a test-piece, turned to a sectional area of one square inch for a length of five inches between shoulders, and provided with large round heads, which were fitted to what is called the "Wade grip." Measurements of elongation, accurate to $\frac{1}{10000}$ of an inch, were made by a special apparatus containing two micrometer screws, with electric contact attachment. The machine was operated by hand-power. Readings of elongation were taken at each increment of 2000 pounds stress per square inch, up to 44,000 pounds, the elongation corresponding to the latter figure being .0088 inch. The elongations up to this point remained almost exactly proportional to the stresses, the strain diagram, as shown on the Plate, from the origin to the point A, being a straight line (the deviation of no observation from the straight line being greater than .0002 inch). There being no indication of an approach to the elastic limit, the attendant at the hand-wheel and scale-beam was told to proceed as usual and move the poise forward 2000 pounds, or to 46,000, and turn the wheel until the beam was balanced.

Noticing that the beam did not rise in the usual time, and that the elongation was increasing rapidly, the writer called to the attendant to stop turning the hand-wheel and to push the poise backwards until the scale-beam balanced. Instead of balancing at between 44,000 and 46,000 pounds, as was expected, it balanced at 40,000 pounds, and the elongation had increased to .0343 inch (or more than four times that recorded at 44,000 pounds), as plotted at the point B, the dotted line between A and B showing the supposed course of the strain diagram between these two points. Immediately upon recording the elongation the writer said to Mr. Gill, who was standing alongside: "This phenomena is without a parallel in my experience, and I know of but one way to account for it. The piece must actually be broken in its interior, through a flaw or weak portion of the metal. The test-piece is now equivalent to a tube, with

a hole in its centre." On further reflection it was suggested, that if this hypothesis were true, it would be confirmed by a comparison of the present modulus of elasticity of the piece with that it had at the beginning of the test. This comparison was made, after first continuing the test till the elongation was increased to .0556 inch, as shown in the diagram, four tests of the decrease of resistance with increasing elongation by resting under stress being made meanwhile, as shown by those portions of the diagram marked, "rested ten minutes," etc. At each of these "rests" the scale-beam was balanced and the elongation read. The piece being left under strain, its elongation increased about .0010 to .0025 inch, in ten or fifteen minutes, while the resistance, as measured by the scale-beam, decreased from 1000 to 2000 pounds.

When the elongation had increased to .0556 inch, with the corresponding resistance, 39,000 pounds, the hand-wheel was turned slowly backwards, relieving the stress, and as the elongations *decreased* the readings were taken at every successive decrement of stress of 1000 or 2000 pounds. The strain diagram of this portion of the test is the inclined line marked "down," the dots to the *right* being the true position of each observation. When the stress had reached 0, the elongation was .0465 inch, a decrease of .0091 inch from that recorded at 39,000 pounds. The hand-wheel was then turned forwards, slowly increasing the stress to 36,000 pounds, readings of elongation being taken at every 2000 pounds. The diagram of this portion of the test is the line marked "up" (it being shifted to the right for the sake of clearness), and the dots to the *left* being the true position of each observation. It will be seen that while the "down" and "up" lines are parallel, the elongations corresponding to any stress on the "down" portion of the test, or while the stress was decreasing, are greater than the elongations corresponding to the "up" portion of the test, or while the stress was increasing. If the strain diagram were accurately drawn it would be a loop, with a maximum width (at 20,000 pounds) of .0004 inch. The observations of decreasing and increasing stress between 0 and 36,000 pounds were repeated three times more, for the purpose of accurately determining the coefficient of elasticity, the most careful readings giving .0080 inch for 36,000 pounds per square inch strain, and the corresponding coefficient of elasticity being 22,500,000, as found by the formula

$$E = \frac{\text{Total stress} \times \text{length of specimen}}{\text{Elongation} \times \text{sectional area}} = \frac{36{,}000 \times 5}{.0080 \times 1} = 22{,}500{,}000$$

The coefficient, at the beginning of the test, when an elongation of only .0088 inch was produced by a stress of 44,000 pounds, was 25,000,000; the decrease of the coefficient of elasticity thus confirmed the supposition that the piece was ruptured in its interior.

Midnight having arrived when the test had proceeded thus far, the specimen was taken out of the machine and laid aside. An opportunity to continue the test did not occur till ten days afterwards. During these ten days the unbroken test-piece was shown to a large number of persons, who were told its history. There was no outward manifestation of fracture, nor any deformation or other sign to indicate that it had ever been under strain. At the end of this time the test was continued, and the piece was then gradually strained from 0 to 36,000 pounds; but to our surprise the elongation was only .0071 inch, and the corresponding coefficient of elasticity 25,352,000, or more than it was at the beginning of the original test. This indicates the existence of a heretofore undiscovered law, viz., the increase of the coefficient of elasticity by resting free from stress after having been strained. The writer, however, does not wish to be accused of generalizing from insufficient data, an error frequently and justly charged against many experimenters, but merely offers it as a suggestion, hoping that others may be led to repeat the experiment and prove whether the law exists or not. After releasing and straining four times between 0 and 36,000 pounds, the coefficient each time decreased until it became 23,228,000. (Query: Does this indicate a decrease of the coefficient due to repeated straining? If so, why was not this shown in the similar test ten days before?) On continuing the test a sharply defined elastic limit was found at 41,000 pounds. There having been no elastic limit found short of 44,000 pounds at the beginning of the test, this is a second confirmation of the hypothesis of a rupture in the interior. The test was then continued till a strain of 70,000 pounds was reached, the elongation being about 0.25 inch, and the piece was left in the testing machine over night under strain. The next morning the resistance was found to have reduced to 66,400 pounds. After releasing the stress to 0, the test was begun again and finished without intermission. An elastic limit was reached about 64,000 pounds, but its position was uncertain, as seen by the diagram; in marked contrast to the sharply-marked elastic limit found at 41,000 pounds the day before. The diagram is a straight line from 0 to 64,000 pounds, but the coefficient of elasticity has again remarkably decreased, and is only 19,380,000. At 73,000 pounds a repetition of the first

curious phenomenon recorded above took place. After balancing at 73,000 pounds, the beam suddenly refused to balance at anything above 70,000 pounds. This is shown on the diagram between C and D. After passing this point, however, the diagram rapidly rises again to 73,000 pounds, then rises very slowly to 75,000 pounds, when the piece broke. Its total elongation in 5 inches was 0.76 inch, or 15.2 per cent. After rupture, a flaw, or appearance of a flaw, about $\frac{3}{8}$ inch in diameter, was found in the centre of the fracture, as was originally predicted eleven days previously. I have brought the fractured pieces with me, that members may judge for themselves of the character of the fracture.

In conclusion, I may say a word concerning Mr. Gill's testing machine, upon which the experiment above described was made. After an experience with nearly all of the common forms of testing machines, and fully aware of both their defects and their excellencies, I do not hesitate to say that Mr. Gill's machine surpasses them all, both in pulling the specimens accurately in line and in facility of operation.

ACCIDENTS IN THE COMSTOCK MINES AND THEIR RELATION TO DEEP MINING.

BY JOHN A. CHURCH, MINING ENGINEER, NEW YORK CITY.

EARLY in the month of August, 1877, a miner in Gold Hill, Nevada, made the unlucky remark that, according to his observation, that month was usually quite free from accidents in the mines. Never was presage wider from the truth. When the month closed twelve mishaps had occurred, killing six and wounding nine persons, and for the rest of that year the Comstock mining communities were kept in a ferment by the frequent occurrence of appalling disasters. There was a real tidal wave of calamity sweeping over the mines, which has not been repeated since, and according to common report was never known before on the Comstock. I was much interested in studying the character of these misfortunes, which also aroused concerted action on the part of the miners, and I have collected a list of the accidents which took place from July, 1877, to May, 1879. The number in this period of twenty-two months

is 101, killing immediately 53 persons and wounding 70 others. This list is incomplete in every way. All accidents which did not injure persons are omitted, and there were several which were of a threatening kind but fortunately did no damage to human beings. Probably others have been overlooked which did belong to the category included in my lists. Finally, no effort has been made to ascertain how many of the wounded died of their wounds, and the number of fatal casualties includes only those who died so soon after their injury that their death became a part of the current records of the accident.

Many of these accidents belong to the usual classes of mishaps in mines and will receive no extended discussion. Others are worth examination from their connection with labor in hot mines, deep mining, and other causes intimately dependent on the local conditions of the Comstock. They may all be classified under eight general heads: 1. Falls of rock, timber, etc.; 2. Tramming; 3. Effects of heat; 4. Falls of men; 5. Explosions; 6. Hoisting apparatus; 7. Overwinding; 8. Miscellaneous. The distribution among these groups in each year is shown in the following summary:

	1877. 6 mos.	1878.	1879. 4 mos.
(1.) Casualties due to falls of rock, ice, timber, etc.			
Number of occurrences,	8	7	8
Casualties, fatal,	2	4	3
Casualties, not fatal,	7	4	5
Proportion fatal, 36 per cent.			
(2) Casualties in tramming.			
Number of occurrences,	4	3	
Casualties, fatal,	1	0	
Casualties, not fatal,	4	3	
Proportion fatal, 12½ per cent.			
(3.) Casualties due to heat.			
Scalding:			
Number of occurrences,	1	2	
Casualties, fatal,	1	1	
Casualties, not fatal,		1	
Proportion fatal, 33 per cent.			
Overheating:			
Number of occurrences,	1	8	
Casualties, fatal,	1	7	
Casualties, not fatal,		3	
Proportion fatal, 73 per cent.			
(4.) Casualties due to falls of men.			
Number of occurrences,	7	7	2
Casualties, fatal,	2	6	2
Casualties, not fatal,	5	1	
Proportion fatal, 62½ per cent.			

	1877. 6 mos.	1878.	1879. 4 mos.
(5.) Casualties due to explosions.			
Number of occurrences,	5	6	3
Casualties, fatal,	3	2	5
Casualties, not fatal,	5	6	
Proportion fatal, 71 per cent.			
(6.) Casualties in hoisting.			
Number of occurrences,	4	6	1
Casualties, fatal,	2	3	1
Casualties, not fatal,	2	6	
Proportion fatal, 43 per cent.			
(7.) Casualties by overwinding.			
Number of occurrences,	1	2	
Casualties, fatal,	1	2	
Casualties, not fatal,	1	1	
Proportion fatal, 60 per cent.			
(8.) Casualties due to miscellaneous causes.			
Number of occurrences,	7	8	
Casualties, fatal,	1	3	
Casualties, not fatal,	10	6	
Proportion fatal, 20 per cent.			
Total occurrences,	38	49	14
Total fatal casualties,	14	28	11
Total casualties, not fatal,	34	31	5
Grand total for 22 months:			
Occurrences,	101		
Total casualties,	53		
Casualties, not fatal,	70		
Proportion fatal, 43 per cent.			

This list is not offered as a summary of accidents in the Comstock mines, but merely as an index to the causes which operate there to endanger life. Many of these are common to all mining operations, but others are quite peculiar and deserve timely discussion, for it is supposed that they are likely to increase in force with the deepening of the mines, which is now rapidly progressing.

(3.) In the third class, or those casualties which are due to the high temperature of the mines and the rock in which they are opened, we have some of the most singular occurrences known in mining. The injuries by scalding were occasioned entirely by falling into the hot mine waters. Their temperature varies with the locality, but the maximum which I have observed is 156° F., and usually it is considerably below this. This temperature seems to be sufficient to produce serious effects. One miner, who slipped into the Julia water, sinking nearly to his knees, got out so quickly that the water did not have time to enter his shoes, and yet his legs were scalded so severely that the skin came off. The same mine was flooded

early in the present year with water which was reported to have a temperature of 158° F., and a miner who in a fit of absentmindedness stepped into it, immersing himself to the chin, was fatally injured. The water is hot and gaseous, and the unfortunate man who falls in it sinks deeply and probably finds it difficult to regain the surface.

But it is in the effects of work in hot air upon the human frame that the most remarkable casualties are witnessed, and this class is purposely put first among those which are to be discussed, because it seems probable that a considerable proportion of all kinds of accidents in these mines are indirectly due to the heat. The proportion of fatal casualties is larger in this class than in any other, being seventy-three per cent., and from the peculiar mental effects of the heat it is obvious that it may be and probably is the initiating cause of many mishaps which would, under other circumstances, be ascribed to culpable blundering.

On the 1900 level of the Gould and Curry mine a drift was run to the southward from the shaft, following the line of the black dike and lying quite near it. This proved to be one of those hot spots which I have before described* to the Institute as a marked peculiarity of the Comstock rocks. As a rule the drifts at this depth have not been above 108° or 110°, and many have been less hot, but this drift has several times been reported to show a temperature of 123°, 126°, and 128° F. Thomas Brown, a miner working in this place fainted, and when taken to the surface and revived was found to have completely lost his memory. He could not tell his name nor where he lived, and had to be dressed and taken home by his friends. The paper which records the occurrence says: "This sudden loss of memory from overheating is quite common in the mines, but the effect soon disappears and the men are themselves again. This fact furnishes an explanation of how men who are considered experienced miners walk off into fatal winzes and chutes, seemingly with deliberate intention."

A frequent accident in these mines is fainting in the shaft while the cage is rising to the surface. The faintness is always felt immediately upon reaching the cooler air a hundred or a hundred and fifty feet from the surface, where there is usually a side draft through some adit. This casualty is so common that a man who has been working in a hot drift is not allowed to go up alone. Long habitude to the heat is no safeguard against this danger, and serious accidents

* Vol. vii, p. 45.

have occurred in this way. The faintness I believe is preceded by nausea, but insensibility follows quite suddenly.

Among the minor casualties I have included one which is said to have happened to Mr. Sutro. Being in the Sutro tunnel before it made connection with the Savage mine, and in an air temperature of 110° F., he went to the air pipe to cool off, and staid there so long that the miners told him to get away from the pipe and let them have air. He did not move, and the account says they tried to stir him up with the handles of their shovels, but he had lost all volition and could not budge. Finally he was put on a car and taken out.

These are the minor effects of the heat. Its graver results are well shown in the following cases of insanity and death:

The first of these, which I introduce with some hesitation, is described by the Virginia *Evening Chronicle* as follows: "At half past nine o'clock Monday morning, March 11th, 1878, a man died at the Caledonia mine in Gold Hill, under peculiar circumstances. The man had been idle for six months, and was working his first shift in that time, he having gone down this morning at seven o'clock. He was put to work as carman on the 1400 level. At the hour stated he rushed into the station at the 1400 level and told the station-tender that the wheels of his car were smashed all to pieces. The station-tender walked back with him to the car, when it was found all right. The station-tender thereupon saw that something was wrong with the man, and took him to the cooling-off place. There he soon began talking wildly and behaving boisterously, and giving other indications of mental aberration. It was therefore thought best to bring him to the surface. He was firmly lashed to the cage and hoisted up, but on reaching the surface he fainted away and died in a few minutes."

The heat on the 1400 level of the Caledonia is not very great, being about 90° F., and this man's sudden decease may have been due to other causes.

Of the other fatal casualties one was from cramps, which the account attributes to the heat, but which may have been the result of drinking ice-water, and another was from a cold taken while cooling off after being partially overcome by the heat. It is to the drinking of ice-water and the comfort of a strong draft that the men resort for recovery from the exhaustion caused by the heat, and though these methods, so contrary to the ordinary rules of hygiene, are put in use several thousand times a day, and usually with impunity,

these two instances are proofs that they may be dangerous. The miners consider the draft of cool air safer than drinking copiously of ice-water, but when in the mines I chose the latter method to save time, and never felt ill effects from it.

The next case illustrates the violent effects which excessive heat may have upon a person not accustomed to it: "On Friday, October 11, 1878, John McCauley went to work for the first time in the Imperial mine. He was cautioned against overexerting himself in the extreme heat of the lower levels. He replied that he thought he was strong enough to stand anything and paid no attention to the advice. At half-past two in the afternoon he was brought to the surface in an unconscious state and died the next morning at half-past ten o'clock."

Two other cases very similar to this have occurred in the Imperial within a few years. This mine is excavated in one of the hot spots of the Comstock.

The hot drift on the 1900 level of the Gould and Curry is the scene of the most serious of these casualties due to heat. Five men were sent there in June, 1878, to load a donkey pump on a car. The work was so exhausting that when the pump caught on a plank they were not able to move it. They seem to have been in a state of mental confusion, but felt that they could not remain longer. Starting up a winze which connects with the 1700 level one man fell on the way, and the others were afraid to stop to help him, but pressed on, reaching the 1700 in half an hour from the time they left it. They were very confused and nearly speechless, and hardly realized what had occurred. Three men went down to the rescue, and found the fallen man still alive. Clearing the pump they got into the car and signalled to hoist, but on the way up the winze the man they had gone to rescue reeled and fell off. The car was stopped at once, but he was jammed between it and the brattice so fast that the others left him and went for help. They all gave out, two half way up, and the other just as he reached the 1700 level, where a friendly hand pulled him up. A new rescue party went down and found two men dead, and the third died soon after. The shift boss reports that "the accident was due solely to the heat, as the air is good enough and pure enough, barring the heat." The winze was not an abandoned one, but in daily use. A heavy volume of steam is reported to rise through it from the 1900 level, the temperature of which, at the time of this accident, is given at 128° F. I gather from the detailed account that the death of the men is possibly at-

tributable to the fact that when the miner fell off the car the latter was stopped in a place that was hotter than the rest of the winze.

It is to be regretted that no adequate studies have been made upon the precise physiological phenomena presented by death under these circumstances. The legal requirements are satisfied when it is proved that the casualty was due to "heat," but if the theory of heat production in these rocks, which I have advocated before you,* is correct, and currents of hot gas rich in carbonic acid are pouring through narrow belts of shattered rock, the death of men who are stopped in one of these belts may have a more complex cause.

(4.) The most appalling accident which can occur in mining work, the falling of men down a deep shaft, repeats itself in the Comstock mines with a frequency which I believe is unknown elsewhere. Sixteen occurrences of this kind took place in twenty-two months, and ten of the casualties were fatal. There were seven falls in the shaft, six in winzes and one in a chute, and three in the floors. One of these deaths was traceable to the effects of foul air, the lights having gone out and the party being on the cage in retreat from the drift.

These dreadful accidents being more common on the Comstock than in any other mines that I know of, it is important to ascertain whether this frequency is attributable to the heat. In some cases we may answer in the affirmative. When a timberman repairing the upcast shaft of an unsually hot mine falls to the bottom we may fairly conclude that the tendency of blood to the head which the lifting of a heavy weight occasions may have been increased to momentary stupefaction by the heat, steam, and gases of the shaft. But though my own impression before examining this subject was that the heat was largely accountable for these mishaps, I am forced by a study of the casualties in this record to admit that this is not the most frequent cause of them. If the Comstock miner is more liable than his fellows to this form of accident it is because he is more often called to work in the shafts after they have been completed.

The Comstock shafts are all sunk in the hanging wall, and the vast excavations which have been made in most of the mines make the settling of this wall inevitable. But besides that the Comstock rocks are forever moving, swelling, and forcing the shafts out of line. No shaft there is in perfectly good condition. Some stand re-

* Vol. vii, p. 52.

markably well, and keep in working order for years, while others require frequent repairs; but from the day they are completed their deterioration is steady, until a general overhauling is necessary. They are timbered in a manner which allows remarkable variation from their original alignment without loss of local support to the ground, and this timbering is rapidly and conveniently readjusted when pushed too far out of line. The work of repair is necessarily dangerous and more hazardous in a hot steaming upcast shaft than in a cool one, and it is to the frequency with which the Comstock miner is called upon to perform this work that the number of falls in the shaft is to be attributed. It is also undeniable that the unfavorable heat conditions may contribute essentially to the result, but of the two causes I consider that frequent opportunity is the greater.

But the falling of timbermen is not the only mode in which this accident occurs. Another is the product of pure forgetfulness. A man working near the shaft will sometimes step off upon vacancy and meet his death. The movement is made quietly, not in the heat of action but after the completion of some task, and above ground such actions are attributed to "absentmindedness." I think this state of inactive perception is exceptionally frequent among Comstock miners, and it stands in such strong contrast to the habits of forethoughtfulness to which they train themselves that we must attribute it to the effect of physical exertion in hot air.

Another class of falls in the shaft are partly due to the high professional spirit and sense of individual responsibility which makes the Western miner one of the most trustworthy of his class. This mode of occurrence, which is unexpectedly frequent, is the pushing of a car into a shaft where there is no cage. This is sometimes due to pure absentmindedness, and sometimes to an interruption of the routine work by a cage going up empty, with a message, with tools, or with a passenger. It is said to be a fact that when a carman has pushed his car to the open shaft and has suddenly awakened to the dreadful situation, he never fails to sacrifice his own life in efforts, however hopeless, to stop it. Whenever a car thunders down the shaft there is always a man with it, and whenever the circumstances have come under observation it has been evident that the loss of life was not due to stupefaction but to a dogged determination to stake everything on the hope of preventing a possible calamity to men below. The spirit is well illustrated by the act of a man who fell down a shaft at Bodie, and used his last breath in calling out "Look out below!" Though this casualty appears so frightful

to those who witness it, it is probably as painless as other modes of certain instantaneous death. I have been told that the unfortunate men always lose their shoes in their fall, and also that they invariably reach the bottom sooner than the car, if one falls with the man.

(5.) The history of blast accidents in the Comstock mines enforces the truth that the nitroglycerine explosives do not enjoy the immunity from ignition by friction which has been asserted for them so confidently. We have to admit that a box of cartridges can be thrown off a house with safety, for that has frequently been done, but inclosed in a drill-hole they have been fired repeatedly by friction, jarring, or a blow. Out of fourteen accidents five occurred from the explosion of unsuspected cartridges by one of these causes.

Two cases of wounding are peculiar from the fact that the exploding cartridges had been in the rock a long time. Each of them lay, not in the header, but in the floor of a drift or chamber, and the accident occurred when orders were given to level up the floor. The explosions were in different mines, and one of the miners was using a pick, the other a gad. Neither was killed.

Two men were wounded by the explosion of a cartridge which they were drilling out for repriming. Three accidents occurred from the explosion of old cartridges left in the header, the presence of which was not suspected, and which were fired, not by direct impact of the tool, but by the jar or the crowding in of the rock upon the cartridge owing to the starting of a new drill-hole near by. Two of these were caused by machine drills, and one by hand drilling. It is supposed that in some cases at least the exploding cartridge was not a complete one but merely a fragment, the upper portion having done its work. This seems probable, for instance, in the cases of the two men who were injured while levelling off the floor, for it is hardly conceivable that a full cartridge could blow up the floor beneath a man's feet and leave him alive.

One of the explosions took place under peculiar circumstances which exhibit very well the sensitiveness of nitroglycerine to the conditions in which it is placed. A hole having missed fire the next shift opened and recharged it. After lighting the fuse the men waited the usual time, and one of them then went to the face and found the cartridge was "boiling," making what miners call a "stinker." It was concluded to drown the hole out, and a man took up water from the wet floor in his shovel and threw it on the boiling cartridge. While stooping for another shovelful the half burnt charge exploded. The superintendent, Mr. Forman, who was pres-

ent, attributes the explosion to the confinement of gas by the water, and the suggestion is a sensible one.

(6.) Hoisting on the Comstock has peculiar dangers from the movement of ground in the shafts already spoken of. It is often impossible to keep the guides in anything approaching a straight line. The shove is not only considerable, but it is frequently confined to short reaches in the shaft, and this displacement of line is the cause of frequent dangers and sometimes of serious casualties. The cage sways from side to side with motion sufficient to make it absolutely necessary to hold firmly to a rod placed in the top for the purpose. Among the casualties of 1877 was one caused by the violent surging of the cage in the Consolidated Virginia shaft, by which a miner was thrown off his balance, one foot going over the side. He escaped with a sprained ankle. Sticking of the cage during its descent and its subsequent fall with a jerk, is one of the most frequent causes of these accidents, and the cage sometimes also gives a sudden bound when rising.

(7.) Though overwinding is neither the most frequent source of casualties nor the most fatal in its results, it has aroused more attention than any other. In twenty-two months cages containing men were twice "run into the sheaves," as the local expression is, and I have notes of four others where no men were hoisted, but one of which proved injurious to persons. Two of the fatal cases occurred at the same mine within five and a half months, and the other, which was caused by overwinding a heavy bailing tank, followed soon after. These, and the running of several empty or ore-laden cages into the sheaves, made a great excitement, and the Society of Engineers tried to prove that the men at the engines were overworked. They formerly worked twelve hours at many shafts, and on solicitation were reduced to eight hours at some. The pay for twelve hours' work was usually $6, and for eight hours $4.50 or $5.

One result of the excitement occasioned by these dreadful casualties was the introduction of several safety devices. Nothing new was presented, there being detaching devices which depended on safety catches for supporting the cage when connection with the cable was severed, others with especial chairs to detain the cage, and one with a rope wound on a false reel to support the cable and carry it quietly to the hoisting drum or reel. In addition the usual appliances for throttling the exhaust from the cylinders of the hoisting engine, shutting off steam gradually and putting on the brake have been re-invented. The gallows frames at most of the old

shafts are thirty-five feet high, and with a "double-decker" cage that takes up twelve or fourteen feet of this space, it is evident the engineer has but a small margin for safety. The new shafts have frames forty-five feet high. It seems to me that the disinclination to use preventive applicances should be overcome. The tendency on the Comstock is to employ unbalanced reels, and with these mere bleeding of the steam pipe should be sufficient to prevent serious injury. With balanced reels it would be absolutely necessary to put on the brake also.

(8.) One of the accidents classed among the miscellaneous causes is worthy of attention. Several years ago an electric signalling apparatus was introduced into the Savage mine, but abandoned on account of the uncertainty of its signals, which would sometimes sound without human aid. In 1877 a new apparatus was put in with improvements which it was thought would prevent this disadvantage, but after working well for some time it failed one morning, and caused a fatal casualty by sounding only three bells when ten were intended. Whether the signal man was at fault is doubtful, but it is a fact that the apparatus had worked badly that morning, and before the accident a man had been stationed to signal with the bell rope in case the battery did not work. There is little confidence in the electric mode of signalling, and I believe it is not in use now in any mine on the lode. A small wire hand-rope is used, and on the whole is safe and convenient.

In looking over this list of casualties it is evident that the accidents which possess most interest are those which have near or remote connection with the heat of the mines, for the latter are sinking with great rapidity and becoming hotter each year. During the past eighteen months they have been deepened four hundred to six hundred feet, and it is difficult to deny that the greater depth increases the chances of accident in mines which, in addition to the ordinary liabilities of mining work, have the insidious and ever-present effects of hot air, water, and rock upon the physical and mental condition of the men. While I have never myself doubted the ability of the excellent managers to carry the works in these hot rocks down to the greatest depths that are anticipated for mining in any region, I find there is widespread doubt in the West upon this subject. There is an impression that the Comstock lode has nearly reached the end of the mining rope, and the reason usually given is that life cannot be sustained in the great heat of the lowest workings, which heat the popular mind loves to exaggerate. The record

I present in this paper should be sufficient to dispel such doubts. Out of 101 accidents twelve were directly caused by the heat or hot water. Undoubtedly the falls of men down a shaft have been caused sometimes by sudden exhaustion, due to the heat and steam in upcast shafts, but on looking over the accounts of the accidents of this kind which are included in the above list I cannot find any of them that show a connection with this cause. The casualties positively traceable to the heat are therefore twelve per cent. of the whole. Probably the heat increases the bad effects of powder fumes and natural gases, and by making repairs to the shafts more frequently necessary it adds indirectly to the occasions when disasters may occur. I also confess to the belief, which is not sustained by observations upon specific casualties, that some allowance should be made for a less active mental condition, a dulling of the faculties, and a certain recklessness to which the heat sometimes goads the men. On the other hand the heat makes them more cautious except when under momentary impulses, and I have never seen American miners more careful of themselves than in these mines. On the whole the good and bad effects of the heat seem to nearly balance each other, and I think that an allowance of five per cent. for the casualties indirectly caused by the high heat would be sufficient. The specific cases which I present do not warrant even that allowance. The accidents recorded here relate to a time when the number of miners at work in the district was from three thousand to thirty-five hundred, including top men and underground men. Most of them worked eight hours, but some of those on top had twelve-hour shifts, and probably there was a constant force of nine hundred or one thousand men below.

It is not possible to make a comparison between these casualties and the amount of ore raised, as the practice is in statistics of coal mines, for the reason that only two out of about forty mines were producing much ore throughout the period covered by the list. The comparison can be made for those two, the Consolidated Virginia and California, and is as follows for the year 1878:

		Proportion to tons ore.
Number of accidents,	10,	1 to 25,771
Fatal casualties,	5,	1 to 51,542
Casualties not fatal,	8,	1 to 32,214
Tons of ore extracted, 257,718.		

Of the casualties included in the list for twenty-two months there were in these two mines twenty-five accidents, twelve fatal, and

eighteen not fatal casualties, or twenty-five, twenty-two and a half, and twenty-six per cent. of the whole.

From all these facts I conclude that increasing heat will not debar the Comstock mines from continuing to the greatest depths which are considered practicable in the existing adjustment of mining appliances to the market value of mining products. It is true that an air temperature of 108° F. is common at the 2000 feet level, while 110° and 112° are not infrequently observed, and that a drift which showed a temperature of 123°, 126°, and 128° F., according to different reports, has proved to be the most deadly on the lode. It is also true that the temperature of the rock is increasing, and with it that of the air, and at some depth, for the calculation of which there are absolutely no data, the temperature of this drift may be expected to prevail throughout the lode unless the present conditions change. But this being an *air* temperature it is always possible to mitigate it by artificial means. It is already observable that drifts 1800 feet below the surface may show a much higher heat than others 400 feet below them. High temperature is largely a matter of locality and temporary conditions, and by various expedients it may be confined to its localities, or combated in other ways.

Deep mining, by extending the line of work through the moving hanging wall, and compelling increased repairs to shafts, will perhaps tend to heighten the already great frequency of falls of men.

In addition to the heat, the peculiar mode of timbering in square setts, the almost exclusive use of nitroglycerine powders, the necessity of frequent repairs to shaft timbers, the incessant movement of the rocks through which the shafts are sunk, making accidents in hoisting more than ordinarily frequent, and the necessity of transporting large quantities of rock through narrow gangways entirely by human labor, are the conditions in which mining on the Comstock may be said to suffer rather more than the usual liability to danger. Two of these causes, both connected with the movement of the ground, may be expected to increase with depth. Together with the heat they comprise forty per cent. of the whole number of accidents, and we may therefore sum up our conclusions by saying that the conditions of deep mining will increase forty per cent. of the causes which lead to casualties and will leave sixty per cent. unaffected. What the amount of this increase will be cannot be foretold, and indeed cannot be estimated until an ore body has been found and worked at great depth. So far the year 1879, which is pre-eminently one of deep sinking on the Comstock, has been remarkably free from casualties.

The administration of the mines is excellent, and the oversight ample. Though there is no declared law I observed that an accident was likely to be followed by the discharge of some miners, and several times the sufferers went before a magistrate and made oath that they were not only entirely innocent of negligence but absolutely ignorant that there was any danger impending, or that there was any one near who could be injured if an accident should take place. The managers seem to have come to the conclusion, which guides the framers of some European mining laws, that accidents will not cease until the damage is assessed on every one in the neighborhood of the casualty, innocent or guilty. The Miner's Union is an active and powerful body, and does not hesitate to express an opinion on the conduct of the mines.

THE HYGIENE OF MINES.

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[NOTE.—This paper was presented at the Pittsburgh meeting in a partially completed form, and I fully expected to obtain, before the period of its publication, both the data and the leisure required for its completion. The hygiene of collieries was to have been separately considered, and the sanitary conditions of various metallurgical industries connected with mining were to receive an extended discussion, while at the same time the points treated in the paper as presented at Pittsburgh were to have more ample illustration from American and European sources. Delay in obtaining expected information, and the continuous pressure of professional work, have prevented hitherto the execution of this plan; and I can now publish only a portion of what I had designed to contribute to the subject. In the meantime this portion, but slightly changed, has been included as a chapter on the Hygiene of Metal Mines in a work entitled *Hygiene and Public Health*, edited by Dr. A. H. Buck, of New York. An excellent chapter on the Hygiene of Coal Mines, contributed to the same work by Mr. Henry C. Sheafer, of Pottsville, affords me an opportunity to supply one of the deficiencies above alluded to. Mr. Sheafer's chapter contains much material in the way of explanations not required by professional readers; and he moreover devotes a considerable space to the explosive gases, falls of roof, and other causes of accidents to life or limb—a branch of the subject not belonging strictly to hygiene. I therefore make extracts only from his treatise.—R. W. R.]

IT is convenient to divide mines, with reference to this subject, into two classes, collieries and metal-mines. Subterranean quarries, rock-salt mines, etc., present no conditions requiring them to be separated from the latter class.

With regard to the hygiene of American collieries (a branch of the subject which I shall not discuss at length), I take the liberty of quoting some excellent observations contained in a recent article by Mr. Henry C. Sheafer, of Pottsville. Mr. Sheafer says:

"The working miner usually devotes his whole life to that occupation. He frequently, perhaps generally, begins at the age of from eight to twelve years as a slate-picker in the breaker—the building in which the coal is prepared for market—where his business is to sit all day, with twenty or thirty companions of about his own age, and pick out fragments of slate from a thin stream of coal constantly flowing past him. The place in which he works is apt to be more or less open and exposed to draughts. His clothing consists of shirt and pantaloons, usually old and ragged; a battered cap and a pair of coarse shoes—the last often omitted in summer. His whole costume, whatever its original color, is soon stained a uniform black by the thick cloud of coal-dust which fills the breaker, filters through his clothing and begrimes his skin, and which forms a large component part of the atmosphere he breathes. As boy and man, his invariable practice at the close of every working day is to wash himself thoroughly from head to foot, a custom to which his hardiness and generally rugged health in early life are to be largely attributed. His diet, as boy and man, is simple. Pork, salt fish, potatoes, and home-made bread are its staple constituents; but when work is good and money sufficient, all the luxuries of the local market are to be found on the miner's table. He learns to smoke and chew tobacco at an early age, has few or no scruples against the use of either malt or alcoholic liquors, and withal grows up to be a lusty, sinewy youth, who seldom troubles the doctors, unless overtaken by one of the numerous accidents to which his own recklessness not less than his somewhat dangerous occupation exposes him. At the age of eighteen or twenty, if he has not previously entered the mine as a driver, or for some other description of boys' work, he goes in as a laborer, becoming in effect, though not in name, an apprentice to a practical miner, with duties so nearly the same as those of his boss, that, for the purposes of this article, they may be considered identical.

"The miner gets to his work shortly before seven o'clock in the morning, if on the day shift, or between five and six in the evening, if on the night shift. He is dressed in flannel shirt, woollen or heavy duck pantaloons, heavy shoes or boots, and usually with a coat thrown loosely over his shoulders. On his head he wears a cap, a slouch hat, or a helmet shaped like a fireman's, but of smaller dimen-

sions. Whatever the headgear, his lamp, a small tin one shaped like a miniature coffee-pot, swings by a hook over the visor; unless the place in which he works is fiery, when he carries a safety-lamp in his hand. His dinner-can and canteen of water or cold tea are swung from a strap passing over his shoulders. Thus equipped he rides down the shaft or the slope, and, if he is lucky enough to catch a train of empty mine-wagons going to his working-place, he rides in, a distance, it may be, of two or three miles from the foot of the shaft. If no wagons are at hand, he walks most of the way through water and slush, taking small account of wet feet, or indeed of wet clothing at any time, though the roof over him may drip all day long. It is an exceptional case if he wears a rubber or oil-cloth suit, even in the wettest places.

"Two miners, or two miners and a laborer, form a gang, and their work is an alternation of exhausting physical labor and intervals of rest. They work with drilling-bar, powder, and pick, getting down the coal and breaking it to a size small enough to handle; with drills, preparing and charging a hole for blasting; with shovels, clearing away the coal and getting it into the mine-cars to be sent to the surface; and then, when a particular job is done, or a blast is to be fired, they repair to the nearest place of safety, and in their overheated condition sit down in the cold, damp draught of the ventilating current to cool off as rapidly as possible. Is it any wonder that rheumatism, consumption, and 'miner's asthma,' are the common ailments among them? In walking to and from his work along the mine gangway, the miner tries to step on the sills on which the track is laid, thus avoiding the hollows worn by the mules' feet between the sills; and as these are laid from two and a half to three and a half feet apart, the effort gives him a long, slow, swinging gait, the head being thrown forward to counterbalance the body. The same posture is found best for traversing the manways and other smaller passages, the long stride being advantageous in picking the way over rough and uncertain ground, while the bent head escapes projections of the roof, and permits the light of the lamp in the miner's cap to fall on the ground at his feet. The habit becomes fixed, and the old miner may always be known by his bent shoulders and swinging stride. That this unnatural compression of the chest cannot but be injurious is evident.

"Among the most laborious of the miner's duties is setting the timbers which support the roof. The gangway, or general passageway of the mine, is usually from seven to ten feet in height, and

about the same in width, seldom falling below these dimensions in American mines, where thick beds of coal are worked and the cars are drawn by mule or locomotive power, though in the thin beds of England and Wales, they are often so small that a man cannot stand upright in them. The gangway timbers, unless the rock and coal are unusually solid, consist of a prop on each side, with a cross-piece uniting them. They are from 10 to 15 inches thick, of length adapted to the dimensions of the gangway, and being of green wood, are correspondingly heavy, weighing from 300 to 500 pounds, according to size. Yet three men are not only expected to set the side-pieces, but to lift the heavy cross-beam into position far above their heads, and fix it there. The work is so hard, performed as it is beyond the brattice which supplies fresh air, in an atmosphere more or less charged with powder-smoke and carbonic acid gas, that, by the time it is done, all three are thoroughly exhausted and overheated, and in most favorable condition for the reception of colds, lung disorders, and rheumatism. If working in a steeply pitching breast,* though the timbers used are not so large, they are quite large enough to tax the strength of the two men who have to get them up a steep and difficult 'manway' by sheer lifting and pulling. In this way, which is almost like working up through a chimney, timbers averaging perhaps eight feet long by six inches thick are carried to the top of the breast, which may be from sixty to eighty yards above the gangway level.

"Mention has been made of the brattice. This is a highly important aid to the ventilation of the mine. It is an air-tight partition, generally carried along one side of the gangway, though sometimes over its top, and so arranged with reference to the ventilating current, that the fresh air is carried along one side of it, while the impure air, which is to be withdrawn, passes along the other. Its object is to keep up a circulation of air in the recess formed by advancing operations at the face of the workings. As every passage or chamber is pushed forward into the solid coal or rock, it necessarily forms a bay, in which the air is always stagnant, unless moved by some such appliance as the brattice. Communicating passages, called headings, are made between the working-chambers about thirty

* In steeply pitching breasts of great thickness—say, like the Mammoth Vein in Pennsylvania—timbers are not used, but the miner works the coal on benches, and works from the top rock downward, having his breast constantly full of coal, on which he stands. In coal-beds of ordinary thickness, say ten feet and less, these timbers are used.

yards apart, for the same purpose; but as the chamber is opened beyond the heading, a brattice becomes necessary here also. One great cause of impurity in the atmosphere in which the miner works is that the brattice is frequently neglected, and the work pushed so far beyond it that it ceases almost entirely to affect the air at the face, which then becomes loaded with powder-smoke and carbonic acid, or, in fiery mines, carburetted hydrogen. In either case the effect on the miner's health is most injurious.

"Of course the principal occupation of the coal-miner is cutting and getting out coal, and here again his work is performed under disadvantageous circumstances as regards the preservation of health. Much of it consists in lying on the side, holing under the mass in a low cut, where every stroke of the pick dislodges a fresh shower of dust, to be inhaled by the miner. Other portions consist of straining at arm's length to dislodge a mass hanging from the roof, of lifting and tugging at heavy weights, of shovelling continuously, hour after hour (where coal has to be shovelled into the mine-cars, the filling of from eight to ten cars, holding three tons each, is considered a day's work for a laborer), and of swinging a heavy sledge in drilling by hand-power. His footing is frequently unsteady, having to be maintained on a steep-pitching floor of smooth slate, so that, as a miner once expressed it to a friend of the writer, 'it is very much like asking a man to stand on the roof of a house while working.' There are chasms under foot and loose rocks overhead, equally to be avoided, and the whole shrouded in a darkness which the miner's lamp reduces only to a semi-obscurity, and which hides without removing the danger.

"The miner's life when not at work also has its effect on his general health, and, as with every other class of men, this varies according to the tastes and temperament of the individual. His house is of frame, plainly but conveniently built, and furnished with the necessary conveniences of life. Being situated in the country, and in a section where land is of little value for either building or agricultural purposes, there is plenty of space about the house, and fresh air in abundance. Even the close neighborhood of frequent hog-pens and occasional stables, and the universal practice of emptying slops from the house on the ground at the back door, have little or no deleterious effect, being neutralized by the abundance of pure air with which their odors and gases mingle.

"The miner's first care on coming from work is to take a tub-bath, cleansing his skin thoroughly. He then dresses in a clean suit,

eats his supper, and is ready for the duties and amusements of the evening, both of which are few and simple. Usually the male inhabitants of the 'patch' gather in groups in the open air, in the village store, or in the omnipresent saloon, and smoke and talk, until the coming of an early bedtime sends them home. Comparatively little drinking is indulged in except on pay day, which comes once a month, and is celebrated by the drinking classes with a 'spree.' In this particular the miner's nationality makes itself seen. While men of all nations may be found drinking to intoxication, the practice as a race is confined to the Irish. There are few of American descent among the miners, and these are generally found among the best and steadiest of their class. The Irish are the most numerous, and they are fond of liquor, drink to excess, and are very quarrelsome when drunk. Terrible fights often accompany a pay-day spree among them. Next to the Irish, in numbers, are the Welsh, a temperate, thrifty, and intelligent race, who form a valuable element in the population. They are industrious and economical, generally succeed in securing homes of their own, which they delight in beautifying and keeping in order, and are apt to be found in positions of trust and authority in later life. Germans and Poles, too, are industrious and economical, but less intelligent and less temperate than the Welsh, more careless in their personal habits, and utterly regardless of the laws of health. They eat unwholesome food, sleep in ill-ventilated rooms, and early acquire a sallow, unhealthy appearance. Nevertheless, their active occupation and the enforced cleanliness of the 'shifting-suit' counteract many of the ill effects of their mode of living, and they will probably be found to average as long lives as the other races. Less numerous, though making up the bulk of the population in certain localities, are Scotch, English, and Italian miners. The last are much like the Irish in habits, while the others hold an intermediate place between them and the Welsh. It is of course to be understood that these remarks apply in general to the nationalities; there are very good workmen and excellent citizens in all classes, and, similarly, there are worthless characters in all; but the general tendency will be found as has been stated. As in every other occupation, personal habits have their effect on the constitution, and predispose it to invite or to repel disease. Thus, drunkenness causes gray tuberculosis, which, with the inhalation of dust and noxious gases, predisposes to consumption, a very common disease in mining towns.

"One of the most prominent conditions of a miner's working life

—certainly the first to be noticed by the casual visitor—is the absence of sunlight, a very deleterious condition as many physicians and engineers of large practical experience consider it, while others as positively deny that it has any injurious effect. Dr. J. T. Carpenter, of Pottsville, in a paper read before the Schuylkill County Medical Society, says (*Transactions Medical Society of Pennsylvania*, 1868–9, p. 488): 'The deprivation of sunlight must be a very strongly predisposing cause of disease. It is to be expected that the results of this deprivation will become apparent in general anæmia, in chronic nervous irritations, in tendencies (easily to be developed by exciting causes) toward scrofula, tubercular phthisis, and allied maladies.' Other practitioners, however, assert that the deprivation of sunlight is among the least of the miner's afflictions; that no injurious effects from it are perceptible, and that no acute disease can be traced either wholly or in part to this cause; while physicians will probably continue to differ forever as to whether or not absence from sunlight during all the working hours predisposes to or prolongs any chronic complaint. In this connection it must be borne in mind that the miner's work is carried on wholly by artificial light, and that usually of a very poor quality. Not the faintest ray of sunlight can penetrate to him, and about the first thing the unaccustomed visitor usually remarks is that it is so *very* dark. It needs but a slight exercise of the imagination to persuade him that he has at last found a sample of that 'thick darkness that might be felt' which once visited the land of Egypt.

"In the winter season, especially when the mines are working full time, their inmates, as a rule, see but little of the sun during their working days. They enter the mine before sunrise, and quit it after sunset. It is, however, a very common practice among them to work week about, one week by day and the next week by night. In this case they have at least from four to six hours of every day's daylight during their night week, and in any case they usually spend Sunday above ground. They do not complain of want of sunshine, and it is difficult to trace any ill effects of its absence upon them. Their complexions are pale, but not more so than those of persons who work at night, or in shaded rooms above ground; and their eyesight, as a general thing, considering the miserable light they have to work by, is remarkably good. Few miners are compelled to wear eye-glasses, for either working or reading, before reaching old age. . . . .

"Too much care cannot be exercised to guard against carbonic

acid gas in mines. It not only exists in large quantities in a natural state, but is constantly being formed by the exhalations from the lungs of men and animals, the products of combustion in the miners' lamps, the ventilating furnaces, and especially the small locomotive engines now so commonly employed. When mixed with common air it is only safe up to the proportion of five per cent., though it is said that some miners become so accustomed to it that they can breathe an atmosphere charged with twenty per cent. of carbonic acid. Mr. Andrew Roy, State Mine Inspector of Ohio (Third Annual Report, 1876), calls special attention to the insidious workings of this unseen but deadly foe of the miner. 'The air,' he says, in speaking of the comparatively shallow mines of Ohio, where natural ventilation is depended on to a very great degree, 'is best in the morning, because the circulation is partially, if not wholly, renewed in the night, during the absence of the miners; but in the afternoon and toward quitting-time it becomes very foul, and miners frequently leave work because their lights will no longer burn, or because they are so oppressed with languor and headache that they can no longer stay in the mine. The black-damp, however, is more insidious than direct in its operations, gradually undermining the constitution and killing the men by inches. By reason of constant habit, young and robust miners are able to stay several hours in a mine after a light goes out for want of fresh air, where a stranger, unused to such scenes, would fall insensible, and, if not speedily removed, would die.'

"Similarly, Mr. J. K. Blackwell, appointed British Commissioner of Mines in 1849, with instructions to make an inspection of their sanitary condition, reports: 'There is another class of injuries, resulting from defective ventilation, to which miners are exposed. The circumstances producing these injuries are slow in operation, and as their effects bring disease, and not immediate and sudden death, their existence has been little considered. These effects are the result of an inadequate supply of air, which has become vitiated and unfit for breathing, on account of its having lost its due proportion of oxygen, which is replaced by the formation of carbonic acid. This gas has its sources in respiration, the lights of the mine, the decomposition of small coal in the goaves (cavities of the roof), and of timber in the workings. Air in this state is also usually found to be loaded with carburetted hydrogen, yielded from the whole coal or in the goaves. Sulphuretted hydrogen, arising from the decomposition of pyrites, is sometimes present, especially in coal-

seams liable to spontaneous ignition. The gases formed by blasting are also allowed to load the air of mines to a very injurious degree.'

"And Thomas E. Foster, Government Inspector in 1864, says: 'In collieries that I alluded to as being badly ventilated, they had no inflammable gas, *and that was the reason why they were not well ventilated*. Although you sometimes kill a few men by an explosion, these collieries where they have no inflammable gas kill the men by inches. There are quite as many, in my opinion, killed where there is nothing but carbonic acid gas, as where there is inflammable gas. The men's health is naturally destroyed, and they kill them by inches. They do not go immediately, but they go in for a few years and die.' Attention is especially called to Mr. Foster's remarks. Colliery managers are altogether too prone to think that fire-damp is the only 'damp' that is to be feared, and force their men to work year after year in an atmosphere loaded with carbonic acid, because in this gas they die slowly and one by one, dropping off without any of the dramatic circumstances attending death by an explosion. It is cause for congratulation that the improved state of science and the requirements of the mining laws in all civilized countries have greatly improved the condition of the mines with regard to ventilation. . . . . .

"Another evil too commonly met with in coal-mines is the cloud of dust with which the air is loaded. Where the coal is kept damp by the percolation of water, little dust is made, and the miner is comparatively free from its injurious effect; but it is exceptional for the coal to be in this condition, and it has been found that the deeper the workings penetrate the less water is found and the drier and more dusty the coal becomes. Any one who has seen a load of coal shot from a cart, or has watched the thick clouds of dust which sometimes envelop the huge coal-breakers of the anthracite region so completely as almost to hide them from sight, can form an idea of the injurious effect upon the health of constant working in such an atmosphere. The wonder is not that men die of clogged-up lungs, but that they manage so long to exist in an atmosphere which seems to contain at least fifty per cent. of solid matter. Ventilation mitigates this evil, but does not obviate it, as a stream of pure water flowing into a muddy pool, of which the bottom is continually being stirred up, will thin the contents of the pool, but will not make them clear. Every fresh stroke of the pick or the hammer, every shovelful of coal moved, every fall of a disloged mass, causes a fresh cloud of dust, until the ventilating current would need to flow with

a force little short of a hurricane to keep the miner's lungs supplied with unvitiated air. Inspector Roy, who has given much attention to the subject of mine ventilation, says (Report for 1876, p. 92): 'Constant labor in a badly aired mine breaks down the constitution and clouds the intellect. The lungs become clogged up from inhaling coal-dust and from breathing noxious air; the body and limbs become stiff and sore, and the mind loses the power of vigorous thought. After six years' labor in a badly ventilated mine—that is, a mine where a man with a good constitution may from habit be able to work every day for several years—the lungs begin to change to a bluish color. After twelve years they are black, and after twenty years they are densely black, not a vestige of natural color remaining, and are little better than carbon itself. The miner dies at thirty-five years of coal-miners' consumption.' Mr. Roy attributes the frequent strikes and other expressions of discontent among the miners primarily to defective ventilation, saying: 'The sources of discontent among miners arise, not, in my judgment, so much in the evil nature of the men, as in the evil genius of the mines; and no conspiracy laws are needed to compel miners to be law-abiding citizens, but better ventilation to expel the demons of the mines—those noxious gases which in remoter ages the priests of Germany were wont to combat with religious exorcisms.' The following cases reported by Dr. William Thomson, show the condition of the lungs above referred to: 'D. C., aged 58; miner for 12 years; lungs uniformly black and of a carbonaceous color. D. D., aged 62; miner from boyhood; lungs uniformly black. G. H., aged 45 years; lungs uniformly deep black through their whole substance, with a density equal to caoutchouc. L. A., aged 54 years; miner all his life; whole lungs dyed with black carbonaceous matter.'

"Dr. R. C. Rathburn, of Middleport, Ohio, testified before the Ohio Mining Commission, on this subject, as follows: 'I have made two post-mortem examinations in which there was carbonaceous solidification in the air-cells. The Scotch people call it spurious melanosis, really a coal-miners' consumption. I have no doubt the carbonaceous particles caused their death. I examined them after death, because before their decease they spit up a black substance whose real character I wished to ascertain. Four cases came to my knowledge.'

"The black substance referred to is solid carbonaceous matter, inhaled while at work. As noted above, it is very slow to operate as a direct cause of death; but aggravates diseases of the lungs, acting

principally as an irritant. Once in the lungs it remains there ever after, manifesting itself in a peculiar black sputum in all cases of expectoration from lung troubles.

"Dr. J. T. Carpenter, of Pottsville, in his treatise before quoted, says: 'I saw, a short time since, a patient suffering from chronic bronchitis, with coal-dust sputa, who had not entered a mine for nineteen years. A gentleman of Pottsville, under my care, is now recovering from pneumonia, with softening and abscess of the lung, who in former years was engaged in mines, but has not habitually entered them for eight years past. During his recent illness the characteristic black sputum was constant.'. . . . . .

"After what has been said, it is evident that the greatest necessity for healthful mining is good ventilation. With air-currents sufficient to carry off noxious gases, powder-smoke, and at least the most of the dust, mining becomes not merely a healthful but an agreeable occupation, notwithstanding all that has been said about its perils and drawbacks. The latter may seem a bold statement to those whose experience in mines is limited to a single visit, but it is the testimony of the great majority of miners, and is confirmed by the well-known fact that men who go from farms and shops to work for a season in the mines rarely go back to the old work. There is something about the comparatively free and easy life of the miner, who is to a great extent his own boss—the uniform temperature, which in most mines varies little, if any, with the seasons, and which ranges from 45° to 65° Fahr., according to local circumstances, the year round—and perhaps the spice of danger which is always present, that makes the miner, once initiated, cling to that work for the rest of his life. Nor is that life necessarily a short one, though the appalling frequency of easily avoidable accidents reduces its average length far below what it should be. So far as the writer is aware, no comparative statistics of the average length of miners' lives, or of their liability to disease, have ever been published; but old men are common among them, and men who have worked thirty, forty, or fifty years in the mines, and are still hale and hearty for their age, are by no means rare. Their principal diseases, as before stated, are miner's asthma, consumption, and rheumatism, and, among those who have worked long in badly ventilated places, dyspepsia, tremors, vertigo, and other troubles arising from blood-poisoning. The two principal causes are dampness and bad air. Pumps and precaution obviate the one, and proper ventilation the other. . . . . .

"In conclusion, it is the opinion of the writer, formed from long

personal acquaintance with the subject, and sustained by the almost unanimous testimony of practicing physicians, mining engineers, colliery owners, and miners themselves, that, were it not for accidental injuries and deaths, the mining class would show as good average health, as fair a percentage of longevity, and as low a death-rate as any other class of manual laborers; that the hygienic conditions of American mines are receiving more attention and consequent improvement year by year, and that, if the average miner could only be taught to exercise caution and common-sense about his work, the list of fatal accidents would be materially shortened, and mining would lose most or all of the terrors which now invest it in the minds of the general public."

Coming now to the second class of mines, I wish to inquire whether the general conclusions expressed by Mr. Sheafer with regard to collieries are equally applicable to metal-mines.

The chief differences in this country between the sanitary conditions of coal-mines and those of metal-mines are the following:

1. The coal-mines are, as a rule, neither very deep nor very high above the sea-level; whereas a large proportion of the metal-mines are situated at great altitudes (5000 to 13,000 feet above tide). The comparative rarity of the atmosphere, though not perhaps injurious to health *per se*, nevertheless intensifies the changes of temperature to which both the mountain climate and the underground work render the miner liable, and thus promotes certain febrile and rheumatic complaints.

2. Although it cannot be said of American metal-mines, in general, that they are deeper than the coal-mines, yet it must be admitted that they grow deep faster, and that the deepest of them far exceed our coal-mines in this respect. In some cases—notably in that of the Comstock Lode—the increase of heat in depth is a very serious inconvenience and injury to the mining work.

3. With rare exceptions, metal-mines do not generate poisonous or explosive gases in large quantity or in brief periods. Slow decomposition in the rocks of minerals such as pyrites may give rise to sulphurous or sulphhydric gases; carbonic acid may be generated by decaying wood, or by the burning candles, or the exhalations of the workmen; but there is no such imminent danger from these sources as threatens the coal-miner, who may be overwhelmed by a sudden irruption and explosion of "fire damp," or drowned in a flood of "black damp." On the other hand, this immunity from sudden catastrophes due to imperfect ventilation leads, in metal-

mines, to a degree of carelessness in this department of mine engineering of which no one would dare to be guilty at a colliery. As a rule, therefore, the air is much worse in metal-mines than in coal-mines. The former are usually left to ventilate themselves, according to aerostatic laws; and when changes of wind or season cause a reversal or stagnation of the ordinary current, the phenomenon is submitted to with a kind of fatalism. Miners say "the air is bad" in this or that level, very much as one would speak in helpless resignation about the weather out of doors. When the heat or foulness of the air at any point actually prevents work, remedies are applied; but so long as it is merely an inconvenience or a slight enhancement of the price per yard of contract work, it is too often neglected, since neglect is not exposed to the death penalty.

4. The greater expense and completely unremunerative character of excavations in rock such as usually incloses metalliferous deposits, leads to the making of much smaller and less regular passages than the gangways of collieries; while separately excavated airways may be said not to exist in metal-mines at all—a brattice or an air-box or a weather-door now and then being the most that is done for the artificial direction of the ventilating current. The smallness of the excavations in metal-mines is therefore another cause of imperfect ventilation. On the other hand, the old workings, particularly if well-packed with "deads," or waste-rock, do not need to be ventilated so much as is often the case in coal-mines, to prevent the accumulation of dangerous gases in them.

5. There is, as a rule, much more climbing in metal-mines. The miners often descend and ascend through great vertical distances by means of ladders and stairs.

6. It is in a few localities only, apart from the coal regions, that a permanent class of miners exists. Moreover, the hygienic conditions of most American metal-mines are not extreme; and, finally, the effects often attributed to underground conditions, in other countries, may be largely due to other causes, and it may be that better diet, less prolonged and exhaustive labor, more comfortable homes, and more rational habits have to some extent rescued the American miner from the evils which have been supposed to inhere in his avocation.

The points thus suggested will now be briefly reviewed, under the heads of physical exertion, air, and temperature.

Physical Exertion.—The wielding of sledge and pick, the pushing of cars, the wheeling of barrows, and the lifting of heavy rocks and

timbers are forms of exertion which the miner undergoes in common with laborers of many other classes, and which cannot be deemed, apart from the peculiar conditions surrounding them, specially injurious to health, though they are doubtless more or less competent to cause or to aggravate certain organic diseases. The ascent and descent upon ladders may be considered characteristic of this avocation, though it is involved also in the ordinary method of raising bricks and mortar to buildings in process of construction. Here the hod-carrier not only climbs, but climbs frequently, and carries a heavy load—a practice once common in the mines of Mexico and South America, but unknown in this country, from which its cost as well as its inhumanity has excluded it. It is the custom now to use windlasses or hoisting engines even for buildings, when these exceed one or two stories in height; and it must be remembered that the highest buildings come far short of the vertical extension of ordinary mines. The question, how much the health and efficiency of miners are affected by climbing up and down ladders, has been carefully examined. The loss of working-time involved in this method of transit is serious. But the exercise of climbing itself, if taken slowly and with due caution, and if the heated climber is not afterward exposed to a chill, is not generally held to be injurious to healthy and strong men. Added to other enfeebling conditions, it is said to hasten the period of declining strength; and it is an important objection to the use of ladders in deep mines that they necessitate the employment of the younger men in the lower levels, and thus deprive the mine, at the points where skilled labor is most desirable, of the services of the oldest and most experienced workmen. Ladders placed at a proper angle are better than stairs, since they permit the arms to take part in raising the body. The loss of time and the waste of strength involved in ladder-climbing are shown by the relative amount of work done per man in the upper and lower levels, this amount being, for instance, in the lead-mines of the north of England, one-fifth greater in the upper levels. On the question of health, it may here be added that sailors are not reported to suffer from climbing any more than bricklayers do; and the sum of the whole discussion appears to be, that the economical view of the subject of climbing in mines is more important than the sanitary one.

This view has led to the introduction of the man-engine, and the practice of lowering and raising workmen in skips and cages. This is not the place for a criticism of the comparative merits of these

devices. It is sufficient to say that in most of those American mines which are deep enough to render the use of ladders a matter of hygienic importance, the workmen are lowered and raised by the machinery that hoists the ore; and the ladders being kept merely as a means of transit between neighboring levels, or as a resort in case of accident, do not enter into the hygienic problem.

Air.—The most thorough and satisfactory reports on the air of metal-mines are those of Dr. R. Angus Smith and Dr. A. J. Bernays, included in Appendix B to the Report of the Commissioners, appointed to inquire into the condition of the metal-mines of Great Britain, with reference to the health and safety of the persons employed in such mines. (London, 1864.) Dr. Smith begins with an elaborate discussion of the normal amount of oxygen and carbonic acid in pure air, and, after citing many analyses of distinguished chemists, adopts 20.9* parts by volume of oxygen, and 0.04 of carbonic acid as a fair outdoor average, and shows that in confined spaces, and under various influences, the amount of carbonic acid may be increased indefinitely. At 11 P.M., in the pit of a London theatre, it was 0.32. But many samples of bad air, taken from mines, have shown over two per cent. of carbonic acid. By a series of most interesting experiments, conducted in a hermetically closed lead chamber, containing 170 cubic feet of air, Dr. Smith established, among other important results, the following:

A person shut up in the chamber for five hours raised the amount of carbonic acid to 2.25 per cent. In this atmosphere the breathing was changed from 16 inspirations per minute to 22, and the pulse fell from 76 to 55, becoming, at the same time, so weak that it was difficult to find. On another occasion, when the carbonic acid had risen to 3.9 per cent., the number of inspirations advanced to 26, and the pulse became so weak as to cause alarm. This is a symptom of poisoning by carbonic acid. An experiment, tried by blowing carbonic acid into fresh air, containing 20.1 oxygen, without removing the oxygen, showed that the pulse of the subject was weakened, though breathing was not very difficult, and the candles burned moderately well. Four miners' candles, inclosed in the chamber, ceased to burn at the end of five hours, having raised the temperature from 50° Fahrenheit to 65°, and vitiated the air until it contained 18.8 oxygen and 2.28 carbonic acid. It follows that men can live where candles will not burn; but that the poisonous

* The proportions given throughout this paper are parts in 100, by volume.

effect of carbonic acid begins before its subject is conscious of serious inconvenience. Moreover, it appears that the presence of carbonic acid is a more noxious agency than the mere diminution of oxygen in an otherwise pure air. According to Dr. Smith's experiments, respiration is not affected sensibly by a small or even a considerable diminution of oxygen, when the place of that gas is not taken up by others of a harmful character. But we do not usually have to deal, in mines, with simply rarefied or deoxygenated air. The abstraction of oxygen is due to processes which load the air with such gaseous products as carbonic acid. The facility with which water absorbs certain percentages of its weight of carbonic acid and other gases, explains the fact that the air is more tolerable in wet than in dry workings. Trickling streams or spray perceptibly improve the ventilation; and this means is occasionally resorted to for enabling men to continue work where it would otherwise be difficult.

Dr. Bernays points out another most important fact, namely, that there is a great difference in the personal sensations of comfort or distress occasioned by breathing different atmospheres containing practically the same proportion of carbonic acid. This is undoubtedly the effect of organic impurities, which greatly aggravates that of the carbonic acid. A much larger proportion of the latter may be breathed with impunity when it is the result of inorganic processes, and particularly of the slow oxidation of coal, than when it proceeds from animal exhalations, and the quick, smoky combustion of candles. Dr. Bernays says that he has often found the air of a crowded room intolerable, though it contained not more than 0.1 per cent. of carbonic acid. He mentions also, as a curious fact, that a man may continue to breathe without distress in a confined space so long as it is contaminated by his own breath only, though he could not, without great disgust, enter an atmosphere rendered equally foul by the respiration of others. But I suspect that the inference he suggests is not well founded. It is, perhaps, not the source of the contamination, but the entrance of the observer from purer air, that makes it more repulsive in the latter case.

Carbonic acid and accumulations of organic impurities are most troublesome at the ends of galleries, or in confined stopes, winzes, etc., which are not swept by the general current of ventilation. The operation of blasting in such places has the good effect of breaking up the stagnation of the air; but, on the other hand, it contributes certain impurities of its own, partly volatile, and partly in fine, suspended floating particles. Carbonic acid, sulphuretted hydrogen,

sulphide and nitrite of potassium, etc., are among the products of explosion from ordinary gunpowder. Gun-cotton is less harmful in this respect, and was recommended by the British Commission; but it has never found general application in mines, perhaps because its use in mines, as a quick and violent explosive, has been superseded, or rather forestalled, by the various nitroglycerin compounds. It is well known that the gases from these produce most distressing headache; but this appears to be the effect on those persons only who are unaccustomed to them. I have seen miners return to a stope almost immediately after a blast of dynamite, apparently without inconvenience. This was, however, in a well-ventilated mine. With all explosives it is necessary and customary to allow the gases to clear away before resuming work.

Sulphuretted and arsenuretted hydrogen may be given off by rocks which contain such minerals as pyrites of iron or copper, mispickel, etc., which undergo decomposition in the presence of air and moisture. To this cause, in part, may be due the alleged unhealthiness of the copper-mines of Cornwall as compared with the tin-mines, in which the ore, being already an oxide, suffers, upon exposure, no chemical change.

Besides the gaseous impurities of the air, the dust produced by drilling has been considered a source of disease. This is probably not a serious evil. The almost invariable practice is to put water in the bore-holes to facilitate the work, and there is from this source little or no dust to be inhaled. What has sometimes been mistaken for mineral dust in post-mortem examinations of the lungs of miners, is finely divided carbon; and this is almost certainly attributable, not so much to the occasional inhalation of gunpowder vapors as to the constant breathing of the products of the imperfect combustion of candles. Some reported cases of the "lead-colic" among lead-miners in Great Britain, and similar cases in the "carbonate" lead-mines of our West may be attributed to the inhalation of plumbiferous dust.

The effect of all these impurities of the air has been found on the continent of Europe and in Great Britain to be a peculiar form of "asthma," "consumption," or "anæmia," known as the miners' disease. It is difficult to say how much the general low tone of vitality due to insufficiency of animal food, lack of healthy dwellings, and reckless personal habits, contributes to the prevalence of this disease; but it is probably fair to conclude that these causes weaken the ability of the workman to resist the effects of impure mine-air.

Temperature.—There is a gradual increase of temperature in the rocks of the earth's crust, below the zone of uniform temperature which is found near the surface. The law of this increase in temperature is not clearly established. It is certainly much affected by the chemical reactions which may go on in the rock. Mr. Robert Hunt, in his testimony before the British Commission, says that whatever may be the temperature of the atmosphere on the surface of the earth, there is in the Cornish mines a constant temperature throughout the year at the depth of about 150 feet. Below that point, he says, the increase is one degree Fahrenheit for every 50 feet down to about 750 feet; then one degree in every 75 feet down to about 1350 feet; and below that about one degree to 85 feet. Mr. Henwood (quoted by Prof. J. A. Church, in his paper published in the previous volume of *Transactions* on the Heat of the Comstock Mines) gives for different kinds of rock the following distances in feet, corresponding with each rise of one degree: granite, 51; slate, 37.2; cross veins, 40.8; lodes, 40.2; tin lodes, 40.8; tin and copper lodes, 39.6; copper lodes, 38.4. These figures show how great is the variation due to local causes. Assuming the increase in granite to be least affected in this way, and applying also Mr. Hunt's formula for the rate of increase, we may adopt as a probable standard of comparison a scale of depths and rock temperatures, as follows:

Depth—Feet.	Temperature of rock.
150,	60°
300,	62°
600,	66°
1350,	76°
2000,	84°

It will be generally admitted that most mines are hotter than this, the fact being that the heat given off by lights, explosives, animals, and men is not immediately removed by the ventilation, and hence the rock is perceptibly cooler than the air. But chemical reactions and hot springs in the rock may very greatly raise its temperature; and when this is the case the miners, finding that the rock feels hot, in comparison with the air, say that the lode or the wall "makes heat." Even when the air is still somewhat the warmer, the rock may seem to be so when touched with the hand.

One of the United mines in Wales is mentioned by Prof. Church, in the paper already cited, as possessing springs which discharge water at 116° Fahr., the depth being 1320 feet. The heat of the

air in the workings is given at 100° to 113° Fahr. The hottest mine in Cornwall is, or was in 1862, the Wheal Clifford, concerning which the superintendent, John Richards, testified that the temperature was 102° fifty-one feet below the 1200 feet level, and a "pretty deal hotter" (120° he guessed) at the 1380 feet level. At one time, in a confined working, the temperature was known to rise as high as 128°. Mr. Robert Hunt, speaking apparently of the same mine, says that, by his personal measurement, the air showed 110° in the deep level, and that tests of the rock, made by leaving a thermometer for two hours in a bore-hole, gave from 112° to 114°. He reports the maximum with which he was acquainted as 117°. Mr. Richards says the workmen can endure 120° perhaps half an hour, but cannot continue to work for an hour at 102°, while they can make a four-hour shift without interruption at 95°. Mr. Hunt gives the average time of working at twenty minutes, and says that, on retreating, the men washed themselves in water at 90°, to cool off. Six sets of men were employed, so that each set had one hour and forty minutes to recover from the effects of the twenty minutes' exertion. Four turns of twenty minutes, thus distributed through an eight-hour shift, constituted a day's work. It is not surprising that, under these circumstances, the labor account was heavy. It is said that three guineas per inch was paid for driving a cross-cut in this mine.

These remarkable statements are even surpassed by the recent experience of the deep mines on the Comstock Lode in Nevada. For many data on this subject, corroborating and completing my own hasty observations and recollections, I am indebted to the paper of Professor John A. Church, already mentioned, and to the unpublished memoranda of that gentleman, generously placed at my disposal. In the lower levels of these mines (say about 2000 feet below the croppings of the Gould and Curry, the usual datum-line) the temperature of the rock is generally about 130°. In freshly opened ground the air usually varies from 108° to 116°; but higher temperatures are not unfrequently reported, as, for instance, 123° in the 1900 feet level of the Gould and Curry. The water, which enters the drift from the lode and the country rock, is, however, often much hotter. The vast body which filled the Savage and the Hale and Norcross mines for many months had the temperature of 154°. But the water, like the rock and the air, varies in this respect in different portions of the mines. The ordinary range of "hot drifts" is 105° to 110° air-temperature. The ventilating current is de-

livered at a temperature of 90° to 95°, which seems to be most conducive to comfort. It is blown upon the men through zinc pipes by means of powerful mechanical blowers. The question of present interest being the effect upon the health of the miners of working under such conditions, further description of the peculiar phenomena of the case will be unnecessary.

Before considering the health of the Comstock miners, it should be noted that by no means all, or even the majority of them, are employed in the hot drifts, and, moreover, that these mines are provided with arrangements which enable every miner to bathe and change his clothing immediately upon emerging from underground.

The diseases of the Comstock miners are mainly typhoid and mountain fever, rheumatism, and erysipelas. There is little or no consumption, bladder, kidney, or liver disease. The superior ventilation (apart from the question of temperature) in the mines, the hearty and abundant diet of the miners, the constant enormous activity of their perspiratory functions, and the personal cleanliness resulting from their daily baths, seem to have abolished among them the disease, supposed elsewhere to be characteristic of their avocation. It is admitted by all observers that they are healthier than their wives and children.

As to the immediate effect of the high temperature upon those who work in them, it must be confessed that, while actually working, the men display apparently undiminished vigor, delivering with seven, eight, or even nine pound hammers, very rapid and effective blows. Perhaps a third of the time is lost in resting and cooling. In very hot drifts, a relief gang is employed; and, in extreme cases, four and even six men to the pick have been found necessary. In the main, however, the rapid progress in the hot drifts is remarkable, and shows that the heat does not greatly lessen the power of work, except by necessitating longer or more frequent rest. At the usual temperature of 108°, three shifts of three men each, working in turns of eight hours, advance three to five feet daily in hard rock. This is so much better than the efficiency reported from the hot lode in Cornwall, that we are led to infer that the method of delivering air to the Comstock drifts affects the temperature and perspiration of the miners in such a manner as to protect them to a large extent from the otherwise distressing action of the heat. My own sensations, as I recall them, in a deep and very hot level of the Crown Point (about 116°, I believe), were not specially uncomfortable on the surface of the body, except when a drop of still hotter water fell

upon me. The principal feeling of distress was internal, and was caused by the inhalation of the scorching air.

The question whether those who labor in such places are permanently injured, is more difficult to decide. One of the physicians at Virginia City has declared that "there is not a sound heart in any man on the lode who has worked in a hot drift for two years." This statement is perhaps too strong, though it is possibly true that many of the miners are organically affected. Yet this appears not to interfere with ordinary and equable work, though it may, perhaps, develop into distinct disease under special strain or excitement. After long work in the hot drifts, the men have a waxen color, and are known as "tallow-faces." Prof. Church noticed some men who, without being lazy, showed unusual care in handling their work, and two or three of them told him that they were "broken down" in hot drifts. In the only instance in which the time required for "breaking down" was given, the workman had been employed underground six years.

The actual effect of the heat on the men is, first, excessive perspiration, and, if this is not removed by evaporation with sufficient rapidity, great faintness. The pulse increases, as is shown in the following interesting data, obtained by Prof. J. D. Whitney and Prof. Church, in the 1800 feet level of the Julia Mine, the drift being about 1200 feet long, and having an air temperature of 108° to 110°, while the air temperature at the station or junction of the drift with the (downcast) shaft was only 84°. The following observations were made:

	Pulse-beats per minute.
Carman, after bringing out car, say 1200 feet,	140
Carman, after resting at station,	64
Carman (another case), after partial rest,	128
Prof. Whitney, after walking through drift,	120
Prof. Whitney, normal rate,	60
Prof. Church, after moving about, without exertion,	88

A case of death is reported as follows: A powerful man, accustomed to hot drifts, returned to work after a rest of three months, and entering the Imperial Mine as carman, pushed his first car to the end of the drift, in the 2000 feet level—say 1000 to 1200 feet—loaded it, and brought it back to the station, where another man was waiting to relieve him. But, instead of taking his turn, he dumped the car and started back without cooling off. He loaded the car again at the end of the drift, and proceeded to return, but

was found a few minutes later hanging senseless to his car, and died, I believe, before he could be got to the surface. Another died in the Imperial incline while that was sinking. Three such deaths in all are reported from this mine, which is an excessively hot one. Sometimes accidental deaths may be the indirect result of the faintness caused by the effect of the heat on the circulation. Thus a man fell down the Imperial (upcast) shaft last year, who was probably overcome by the heat while putting in timbers. In these worst places, strong and healthy young men are employed. Fat men seem to stand the heat best, and, among visitors, women endure it better than men. Some men wilt under the work, and are said to have "no pluck." Drinking habits unfit the miner for this severe test. Unaccustomed men are often unable at first even to reach the end of the drift where they are to work. An intelligent miner told Prof. Church that the first month of such work after a long rest is hard; then comes three months of brisk feeling; and then follows a "dragged-out" sensation.

The underground use of machine-drills, operated by compressed air, is an important aid to ventilation and cooling, since the expansion of the escaping air absorbs much heat from the immediate neighborhood. But when, as in the Comstock, the heat radiated from the whole surface of exposed rock is far in excess of that which men and lights supply, nothing can sensibly reduce it, or mitigate its effects, except abundant mechanical ventilation. This is carried to a large extent in the Comstock mines; and to the fact that, in counteracting the high temperature, the impurities of the air are thus removed, the remarkable good health of the Comstock miners may be partly ascribed. Other causes have been already mentioned, such as the healthy mountain climate, the good food, and the comfortable dwellings. Finally, the fact must not be omitted from consideration that the miners of our Western regions are emigrants, and presumably men of such bodily vigor and health as their adventurous spirit would imply.

Incidental to the question of temperature is the effect of sudden changes of temperature, such as are experienced on coming suddenly from the depths of a mine to the surface. The hygienic conditions here do not differ from those which any similar change of temperature produces; and since they may easily be counteracted by the prudent miner, they need not be set down as sources of disease inherent in his occupation.

Another kindred question relates to the effect of barometric pres-

sure, which varies in mines with the depth of the openings, and also with the changes of the outside weather. The general experience is that high barometric pressure, though it permits a greater inhalation of oxygen with each breath, causes a feeling of distress, and affects the heart unfavorably. Dr. Bernays says that undoubtedly the most injurious, as well as the most unpleasant, condition of mine air is that in which a high temperature is accompanied with excessive barometric pressure and great humidity. The effect of the pressure alone can best be studied in the records of work in highly compressed air, as in the sinking of the caissons for the East River and other bridges. It may be affirmed, as a general rule, that sound men are not permanently injured by it. In ordinary mines the chief sensible effect of the barometric pressure is the variation it may cause in the natural ventilating current. Where the ventilation is wholly or partly artificial, these changes may be controlled. The introduction of compressed and cool air by machinery tends powerfully to reduce to a minimum the humidity of hot mines, and thus (as in the Comstock) to give an atmosphere in which free perspiration, rapidly evaporating, cools and refreshes the body. A comparison of the statements above made, as to the Comstock miners and the miners in the hottest mine of Cornwall, shows how much more can be endured and accomplished by workmen when thus protected from vitiated or over-humid air.

The injurious effect of working under artificial light, instead of sunlight, has been often asserted; but there is no definite proof of it. Where other conditions are wholesome, and the habits of the workmen are regular, this is not likely to have a traceable effect. At all events, it is subordinate to many other causes.

General Conclusions.—The British Commission, to which reference has been made, summed up its voluminous report in a few conclusions and recommendations, the substance of which I quote below, in order to point out how far they are applicable to miners in the United States.

The Commission finds that a large proportion of the diseases affecting miners in the metal-mines is to be ascribed to defective ventilation only. However various the opinions of physicians concerning the causes of the disease so well known under the name of miners' consumption or miners' asthma, there is in one respect a remarkable unanimity among all the experts, namely, that the health of the miner is chiefly affected by the quality of the air in

which he works. This conclusion is emphasized by the results of very wide inquiry on the part of the Commission.

In the coal-mines, where special attention is paid to ventilation, on account of explosive gases, the mortality of miners, apart from accidents, is lower than in the metal-mines. Starting from this significant fact, the Commission recommends that some of the methods of artificial ventilation employed in the former should be more generally introduced into the latter, and favors particularly the use of furnaces in "upcast" shafts, to accelerate the natural current by heating the upward-moving column of vitiated air, and to prevent the stagnation or reversal of the current by change of season or weather.

With reference to other causes of disease, the Commission recommends that every mine be provided with a conveniently situated, separate house, in which the workmen may change and dry their clothes; that boys under fourteen be not permitted to work underground; and that mechanical means be adopted for transporting the miners into and out of the mines. The man-engine is praised; but the system of hoisting the men in skips and cages is also pronounced satisfactory, provided the machinery be properly constructed and carefully tended.

These recommendations are as timely now as they were ten years ago, except that the increasing use of compressed air in mining has furnished an aid to ventilation not then considered. There is no proof that the metal-miners of America are less healthy than other laborers, and there is no need that they should ever become so. In my judgment a wise regard for financial economy alone will cause capitalists to do all that philanthropic considerations would require in dealing with the problem of hygiene in mines—a problem which contains, as the foregoing discussion shows, no fatally insuperable difficulties, and no insoluble mysteries.

PROCEEDINGS

OF THE

MONTREAL MEETING.

SEPTEMBER, 1879.

THE first session of the Institute was held on Tuesday evening, September 16th, in the William Molson Hall, of McGill University, Dr. T. Sterry Hunt, Chairman of the Local Committee of Arrangements, in the chair.

Dr. Hunt introduced Monsieur S. Rivard, Mayor of Montreal, who spoke as follows:

GENTLEMEN: We welcome you with pleasure to our city, and we are proud to find that you have deemed it worthy of your visit.

I understand that your association was founded in 1871 with some thirty courageous members, and that so valuable has it been esteemed, and so attractive was it found, that nearly one thousand have now enrolled their names as members and associates. But we must regard your profession as dating back to the stone age and mingling therewith; how remote that is none of us can tell. Much of civilization is due to the discovery of metals, and the magnitude of our works to their widespread abundance.

If you have not accomplished what the alchemists aimed at, which was to turn the ruder materials into gold, you manage, by your subtle skill, to turn stones into gold value. Europe has long since been dug into and perforated with shafts and galleries in search of metals, and great wealth, and power, and comfort have arisen therefrom; but by an increase of knowledge, and more than European enterprise and energy, America is following rapidly in the race, and by means of natural advantages of a geological character has outstripped the Old World.

We are pleased to see you, gentlemen, as an evidence of the unity of this continent. Maps may have various delineations, but the veins and arteries of commerce and the nerves of science knit all together into one social organism with the same vital functions. We live here on the margin of the world, and can claim but a modest share in the great works of modern times, but such as we have that can interest you we gladly offer for your observation. Our Victoria Bridge may be considered appropriate, for without mining engineers to precede the civil and mechanical, no bridge would be there to-day. In truth, the same may be said of all works of metal, and of all that are executed by tools and machinery made of metal. The Mont

Cenis Tunnel, which I have had the pleasure of seeing and admiring, may be placed in the same category, and also that gigantic work, the Suez Canal, and to which may be added in the future, on our side of the Atlantic, the stupendous cleft which shall sever two Americas while it unites two oceans.

Twenty-five years ago Canada had barely disturbed herself from her tillage and her trade, her pineries and her fishing, but so rapid has been the advance in all matters where high science is brought to bear, as is especially the case in your profession, that we can scarcely separate the possible from the impossible, and it is not unlikely that your presence here may stir us up, as under a new inspiration, to look into our unknown treasures and bring them to the light of day. In the meantime, gentlemen, it shall be our united endeavor to make your time pass as pleasantly as possible while we have the privilege of your companionship.

At the conclusion of the Mayor's address, Dr. Hunt introduced Principal Dawson, who welcomed the Institute on behalf of McGill University. Principal Dawson said:

MR. CHAIRMAN: It is to me a very pleasing duty to welcome this Institute in the name of McGill University, and this more especially that this opening meeting is held in one of our own halls. The visit with which we have been honored is one that we could not claim on the ground of any mining operations in our vicinity. Situated on the flat and non-metalliferous Trenton limestone and Utica shale, at the base of a trappean eminence, which, however beautiful, is barren of useful minerals, Montreal is not, and cannot be a mining town, though it is, and may be to a still greater extent in the future, a manufacturing one. Yet, in three other respects, Montreal has some significance in mining affairs. The capital of its wealthy merchants is largely invested in mines, not always, I fear, too profitably in all the vast region from British Columbia to Nova Scotia. No other Canadian city has done so much in furnishing capital for mining enterprises. Montreal has also, through its university, taken the initiative in the Dominion in providing for the education of mining engineers. In its faculty of applied science it has a department of mining and assaying ably conducted by Dr. Harrington, one of your members and of our alumni, and the graduates of McGill are now represented, both in industrial and educational work, by a number of zealous and able young men employed in mining enterprises on the geological survey and in educational institutions. We have endeavored to teach, as thor-

oughly as possible, the scientific principles on which mining depends, to give some familiarity with methods of manipulation, and to obtain facilities for students to study practical mining in their vacations. The public appreciation of our work has been shown by subscriptions toward the endowment of the school, by the presentation on the part of a lady of this city of an admirable series of mining and metallurgical models, and by donations of specimens to our museum. A third and most important reason is the presence in Montreal of the headquarters of the Geological Survey of Canada, so well known through the labors of Sir William Logan, and now ably conducted by Mr. Selwyn, and which possesses the best collection extant of our mining products. For myself I may, as a geologist, claim some affinity with mining engineers, as at least a preparer of their way. In this respect the working geologist may, I think, claim to have done good service, as well as in leavening the public mind with the idea that science is important to the success of mining enterprises, and that the opening of a mine is not merely a blind venture, or something to be intrusted to a mere practical miner or to a man with a mineral rod. Had the value of practical science been more regarded fewer losses would have been sustained, and less scope would have been afforded to the operations of unprincipled speculators. It is a very trite but true remark that descent is easy, and, as applied to mines, it means that any fool can sink money in the earth, but it requires wisdom and knowledge to bring out a return. In this respect your Institute is an anti-gravitation society, and it is the business of the true mining engineer to bring to the surface out of the depths that which is weighty and valuable, whereas, in unskilful mining, there is an inevitable tendency in capital to gravitate toward the centre of the earth, and never to return. I trust that the members of this Institute will be duly impressed with the moral as well as material responsibility which rests on them in this respect, and that the results of your present meeting will tend greatly to the better knowledge of the beneficent art of making the treasures hidden in the earth accessible and useful to the toiling multitudes on its surface. In the hope that this will be attained, and that your visit will be agreeable to you and beneficial to us, we give you a hearty welcome.

President Eckley B. Coxe replied briefly, on behalf of the Institute, to the cordial words of welcome spoken by the Mayor and Principal Dawson, and then read the following address:

MANY of the members of this association have often, I feel sure from my own personal experience, been asked: What is the object of the American Institute of Mining Engineers, or what is the advantage of belonging to your society, or what do you do? etc. Now, it seems proper upon the occasion of this our first meeting held outside of the United States, that a few moments should be devoted to the consideration of the question: What is the *raison d'être* of the American Institute of Mining Engineers, or what is the vital force that animates it, and which in less than nine years has brought it to its present position both as to numbers and usefulness? I would here remark, by way of parenthesis, that we (that is, the Institute), in calling ourselves *American*, have not appropriated that word in the way that is usually done by us Yankees; for we count among our members many whose homes are on this side of the St. Lawrence, and we draw no invidious distinctions between them and those who reside in the United States. We impose upon them the same duties as upon the latter, and do not let them off as we do the so-called foreign members.

One fact must have struck us all, and that is, that a majority of those who attend and take the most active part in the reading of papers and in the discussions at any meeting are present at almost every one, and that they are not gentlemen of fortune and elegant leisure, but the hardest worked and most busy members of the profession—men to whom attending a meeting means not only an actual outlay of money, but also extra labor before going to and after returning from it, and who are sure to find, upon getting back to their offices, piles of letters and papers requiring immediate attention. And yet there seems to be some mysterious charm about the meetings of the Institute which those who come once within its influence seem ever afterward unable to resist, and to whose power they willingly but inevitably succumb. That, at least, is the experience of many that I now see around me.

The first, and I think by far the most important function of the Institute, and that which really has caused it to grow so quickly and so vigorously, is that it *prevents mental waste* and stimulates men to intellectual activity outside of the mere routine duties of their profession. One of the greatest triumphs of science, one of the points in which modern civilization distinguishes itself most from the ancient, is *the utilization of waste products*—in turning that which formerly was either useless or hurtful to mankind into a source of wealth or a thing of beauty. If he who causes two blades of grass to grow

where one grew before is to be praised, how much more he who con verts some useless and fever-producing swamp, by drainage and cul tivation, into a beautiful meadow? But there are many things in th world which may be wasted or which may be hurtful to the humai race besides the mere materials of which the globe is composed, o the unused products of our industries—they are the fruits of men' experience, the results of their thoughts, and the latent capacity o the brains of trained men for useful work when not occupied b their regular business. This capacity, if not utilized, may lead th possessor of it into occupations which will do no good to himsel or to mankind, or it may be gradually weakened or lost if not em ployed. There is scarcely an engineer, however limited his practic may have been, who has not made some experiment, met and over come some difficulty, or solved some problem that has not yet bee discussed in our technical literature. These may not be of th highest importance, and may have occurred in the practice of dozen of other engineers; but so long as the results obtained have not bee reduced to writing and put on record in some well-known work o periodical that is accessible to the profession, they are generall waste products so far as the world at large is concerned; for the may die with the possessor, or, even when they have been carefull preserved by the original investigator, may be sold as waste pape by some one into whose hands they may fall, and who will be unabl to appreciate their value. Before the organization of our Institute the results of much thought, of great experience, and many experiment lay scattered over this continent in the memories of the managers o foremen of works, in the note-books of engineers, and in the drawing rooms of industrial establishments connected with mining. There wa no centre toward which these fragments were attracted and collecte into a compact whole, accessible to, and of importance to every on Now, the case is different; for the American Institute of Mining En gineers offers to its members, as the analogous societies of other pro fessions do theirs, not only a means of, but also an incentive to util izing this great mass of valuable information, which is constantl accumulating in all parts of the country where mining and metal lurgy now are and have been for the last ten or fifteen years makin such great strides in advance. Under ordinary circumstances, a engineer who has made a series of experiments, which has develope some new law or property of matter, and promises to have a mor or less important bearing upon the future of some branch of mining

would often be deterred from publication. I do not, of course, refer here to experiments undertaken for the purpose of determining some fact to be used by those who have been at the expense of the investigation in their own business; these are professional secrets, belonging to those who have paid for them, and are as sacred as the secrets of the lawyer, the doctor, or the clergyman, and no true engineer would ask or even wish them to be divulged, except with the consent of their owner; but I refer to those experiments which all of us make, and which there is no commercial object in concealing. Now it is generally the case that such investigations do not cover sufficient ground to form the basis of a book, and if they do, the engineer who proposes to publish them, if they require plates to explain them, experiences great difficulty in finding a publisher; for the latter naturally looks to the question of how the book will sell, and what it will cost to get it out, and is not likely to encourage the publication of a work which, although intrinsically very valuable, would be very expensive to set up and illustrate, and of which only a small number of copies would be likely to be sold. The author is then met with those pleasant little suggestions, such as to cutting down the size, leaving out expensive illustrations, making the work more popular and less scientific, advancing the whole or part of the cost, etc., with which some of us are familiar, until, finally, he becomes discouraged and lets the matter drop, finding that the world is unwilling to listen to him in that way unless he pays it to come to hear him; for such a work does not appeal to the general public, but only to a select few who incorporate the data obtained in general treatises on mining, metallurgy, etc., or who use it in their own practice. If, on the other hand, there is not material enough for a book, but only for a short essay, it is difficult for him to find a periodical willing to undertake the publication of it, which will reach all those to whom his article would be of interest; and the same difficulty at to illustrations occurs. In many cases the course of experiments undertaken requires years, perhaps a lifetime, to bring it to a successful conclusion, and in that case it does not furnish matter for a magazine article. If, however, at any time a member of an association such as ours should think that he had discovered some fact or principle which had been theretofore unknown to the world, and which promised important results, at the first meeting thereafter he would call the attention of the members to it in a preliminary note, giving only a general outline of what he had done, and of what he proposed to do, reserving details and conclu-

sions for his final paper. Should he, as is often the case, be simply reinvestigating some problem that had already engaged the attention of others, or should some other member be engaged in the same line of study, it would be brought to his attention at once in the discussion that would take place after the reading of the note, and he would be informed of what had already been accomplished, or at least referred to some authority from which he could obtain such information; many of the members would be likely to give him isolated facts, the results of their own experience, which would aid him to explain points about which he was not yet clear.

If what he was undertaking had been already done, he would be in the same position as a traveller who, when engaged in exploring a, to him, unknown country, should suddenly be informed that it was well known, and had been thoroughly mapped. The purchase of a guide-book or geography which had been recommended to him would convince him of the fact, and he would then employ his time and money in some more profitable enterprise. Thus waste is avoided. If, on the other hand, he should really be engaged on some original work which promised to be of value to all, he would be assured of it by the discussion of the subject, and would pursue his investigations with renewed vigor, feeling certain, in the first place, that it was not labor in vain; and, in the second, that he had placed himself on record by the reading of his note and by its publication in the Transactions of the Institute, so that no one thereafter could dispute his claim to be the original investigator or discoverer, or, at least, to be the first to have announced to the world the fact or law which he had discovered. It is very important that such announcements should be made as soon as possible; first, because the world gets the advantage of the knowledge sooner; and, secondly, because it avoids much bitter and useless discussion afterward as to the original inventor or discoverer.

The existence of such an association, to which the results of such work can be made known, and by which its value can be unofficially, but, at the same time, really and decidedly passed upon, is an incentive, particularly to the younger members of the profession, to make original investigations; to solve or help to solve some problem in mining or metallurgy; to try to interest themselves in something outside of the mere routine duties which their professional positions require of them; for although a student, upon graduating from a technical school, may receive a diploma which will entitle him to call himself a civil, mechanical, or mining engineer, and be acknowledged by

all other engineers as such, yet his position in his profession can only be fixed by what he can show to the world that he has done and is capable of doing; and there is no way, I feel sure, in which a young engineer can show so easily, so surely, and so quickly his real ability and have it recognized as by original work communicated to an association such as ours in notes and papers, and by taking part, at the meetings, in the discussion of subjects with which he is familiar. In this way a waste product is utilized; for the mining engineer, placed, as he frequently is, far from the society of congenial persons, has often much time, which, under ordinary circumstances, would not be employed, and which he now utilizes in preparing a paper for the meetings of the Institute.

The second and, in my opinion, the most important function of the Institute is the bringing together the members of the profession who are scattered far and wide over the whole country, enabling them to become personally acquainted with each other, and thus tending to the formation of a professional standard. Ours is really a lonely profession. The lawyer, the physician, the clergyman, and even the civil and mechanical engineer, reside, in most cases, in more or less thickly populated parts of the country, where they are brought in contact with other members of their professions, and thus are kept to a certain extent *au courant* of what is being done outside of their own special routine of duties. They are frequently called into consultation with each other, have access to a much larger professional literature, and are constantly brought into contact with educated and cultivated people of all kinds. But the mining engineer, except in special cases, must spend most of his time where his mine or his furnace is located, and these are generally situated far from the centres of civilization, their position being regulated by the existence of the ore or the fuel. The nature of mining enterprises is not such as tends to build up large cultivated societies around the works, so that the engineer is generally thrown upon his own resources, having access to no professional books and periodicals but those in his own library, which, in most cases, cannot be very numerous, in consequence of the high prices which must be paid for them. His only associates are, as a rule, his foremen and the workmen, and he is frequently cut off from the society, not only of other members of the profession, but also from that of other educated persons who are familiar with what is going on in the outside world. Until recently there were few if any technical periodicals which were of sufficiently general circulation among mining engineers in this country to serve

as a means of communication between those who were thus isolated, and they seldom met together for consultation, so that each went on working out alone and unaided those problems which his duties required him to solve; those who were in the habit of using books knowing more of what was being done on the other side of the Atlantic than at mines within one hundred miles of them, where the conditions were more or less similar to those at their own homes. In fact, before the organization of the American Institute of Mining Engineers, a majority of well-informed people, of those actually taking part in business and professional life, did not know that there was such a profession as mining engineering; or, if they had by accident heard of it, they had no idea what it was—in fact, I think we may admit that among the profession itself there was no very clearly defined idea of what a mining engineer was. But the great increase in the number of mining engineers, the enormous development in this country of all the industries connected with and dependent upon mining, rendered such a condition of affairs no longer possible; and when, in May, 1871, a few gentlemen met at Wilkes-Barre, Pennsylvania, and organized the American Institute of Mining Engineers, it was at once welcomed by the other members of the profession, and its success was assured from the very beginning. That it supplies a real want is best shown, I think, by its six volumes of *Transactions*, by the records of its meetings, and by its list of members, whose achievements in all branches of mining and the associated industries are well known and recognized in all the civilized countries of the world. There is no doubt that the effect of our meetings, our publications, and of the publicity given to our proceedings has been to raise the standard of the profession of mining engineering, and to make the public appreciate its importance. Now, although the Institute is not like a university, an institution for conferring degrees, and although it does not in any way guarantee the integrity or professional standing of its members, yet the rules that govern the admission of its members although far from stringent are intended, and practically do, prevent unworthy persons from being admitted, except in rare instances. It is not the intention of the rules, as I understand them, to require those who propose a new member, to make such an examination of his capabilities, and to give such references as to his character as would enable the Council to decide upon and to certify to his exact standing; but the door is open to all who take an interest in mining or the sciences allied to it, and who bear a good name. They may enter as members; or, if

not eligible under that head, as associates; they may be engineers, lawyers, doctors, or owners of mining property. The mere joining of the Institute confers no title or degree; each one must thereafter make his position in it for himself. It is a true republic, and its motto is, "To each according to his deserts."

Sometimes we find a man who, before entering the society, was comparatively unknown to the other members, but who, in some quiet corner of the country, has been hard at work; in a few short months he comes to the front and takes rank with the *élite* of the profession; his papers and the part he takes in the discussions giving him not only an American, but a European reputation; the success which attends his *début* as an author spurs him on to new and fruitful fields of research, which the knowledge he has gained at the meetings enables him to cultivate with increased effect. Others who may have joined with too high an idea of their own, or with too small an opinion of the ability of their professional brethren, learn to estimate themselves and others at more nearly their true value.

Some find the atmosphere of the Institute uncongenial, and leave a society where brains, hard work, and good-humor are about the only things that enable a man to take a prominent position in it. The result of bringing together, not as the representatives of rival interests, but socially and informally, the various members of the profession who live far from each other, and being engaged in dissimilar occupations, would, in the ordinary course of events, seldom if ever meet, has been the formation of friendships which are of the strongest character, and which will, in my own case, as in many others, I feel sure, terminate only with life.

Another advantage afforded by the meetings is the opportunity of visiting many of the most important industrial establishments of the country, not only in company with men who have constructed and are managing them, but also with those who are most competent to criticise them.

Under these circumstances, an engineer who is an expert in only one branch of his profession, is enabled to keep himself intelligently *au courant* of what is being done in the parallel and collateral branches, and thus obtains many valuable hints which he can utilize in his own particular department.

Another important and agreeable function of the Institute has been the opening up and maintaining of friendly relations with our professional brethren across the water. This, which was so pleasantly and successfully inaugurated at the Centennial, has been kept

up without intermission, and scarcely any foreign mining engineer who visits this country for the purpose of examining its resources fails to avail himself of the facilities offered by our society; and all of our members who have visited Europe since 1876 have found that the attentions shown to Europeans in America by us have been thoroughly appreciated and returned abroad; and I hope that in this respect the usefulness of the society will increase rather than diminish.

There is another function of the Institute, which is, perhaps, the most agreeable one, and that is the breaking up of our every-day routine work three times a year, and forcing us to lay it down, thus turning our minds into a new channel for about a week at a time. Now, although we all know that attending a meeting means harder work before and after it, yet we always look forward to the next one and backward to those we have attended with the most pleasant feelings; and when we return from one of them, we take up our daily duties with a feeling of having renewed our youth, and being able to do more than if we had not been away. We enjoy the meetings as boys at school do their holidays, and a holiday it really is to most of us. I would, therefore, say that our answer to the question, "What is the *raison d'étre* of the American Institute of Mining Engineers?" may be summed up as follows: Our Institute is useful in preventing the immense amount of valuable information, which is being accumulated daily by its members and others, from being lost, and in placing it in a convenient form at the disposal of the scientific world; in encouraging its members to do original work, and to communicate the results thereof to the public; in raising the standard of the profession, and in causing the outside world to understand what a mining engineer is; in bringing the various members of the profession together, so that they learn to know and appreciate each other; in giving the members an opportunity of visiting, in company with a great number of experts, the various industrial centres of the country, and of inspecting the improvements made from time to time; in maintaining pleasant and profitable relations with foreign engineers; and in making a periodical break in our routine duties, and giving us all a week's holiday every three or four months, during which we can talk over together matters connected with our profession in an informal way; and, finally, in forming, cementing, and maintaining professional friendships, which are, in many cases, the strongest and truest.

Dr. Hunt followed, explaining in a few remarks the work of the

Institute, and showing how it legitimately embraced besides mining proper, chemistry, geology, mineralogy, metallurgy, mechanical engineering, and mining law. Dr. Hunt then resigned the chair to President Coxe, who called on Dr. R. W. Raymond for a paper on the Zinc Deposits of Southern Missouri.

On the conclusion of Dr. Raymond's paper the session was declared adjourned, and an hour was then pleasantly spent in the inspection of the library, models, and collections of the University.

On Wednesday two sessions were held, in the morning and afternoon, in the lecture-room of the American Presbyterian Church, when the following papers were read and discussed:

The Law of Fatigue and Refreshment of Metals, by Professor T. Egleston, of New York.

Washing Phosphoric Pig Iron for the Open-hearth and Puddling Processes at Krupp's Works, Essen, by A. L. Holley, of New York.

Losses in Copper Dressing at Lake Superior, by Professor H. S. Monroe, of New York.

Recent Improvements in Concentration and Amalgamation, by Professor J. A. Church, of New York.

Experiments with Charcoal, Coke, and Anthracite, in the Pine Grove Furnace, by John Birkinbine, of Pine Grove Furnace, Pa.

The Silver Sandstones of Southern Utah, and Cost of Milling Silver Ores in Utah and Nevada, by R. P. Rothwell, of New York.

A New Method of Dredging, applicable to River Mining, by Dr. R. W. Raymond, of New York.

Mr. J. C. Platt, of Waterford, New York, exhibited and described a new journal-bearing.

On Wednesday evening Dr. and Mrs. Hunt entertained with graceful hospitality the members of the Institute, and ladies accompanying them, at their home on McTavish Street.

Thursday morning was devoted to the inspection of the museum of the Geological Survey of Canada, under the guidance of Mr. A. R. C. Selwyn, F.R.S., Director of the Survey, and to a delightful drive through Mount Royal Park and environs.

The fourth and concluding session was held on Thursday afternoon.

Mr. Thomas Macfarlane, of Actonvale, Quebec, Canada, read a

paper on Silver Islet, describing its discovery, working, and production. The following members and associates were then elected:

MEMBERS.

H. W. Armstrong,	Pittsburgh, Pa.
Charles A. Brinley,	Philadelphia.
Henry W. Bulkley,	New York City.
William Hartz,	Johnstown, Pa.
A. Heckscher,	Shenandoah, Pa.
Prof. Charles H. Hitchcock,	Hanover, N. H.
S. Warren Ingersoll,	Philadelphia.
George Jamme,	Londonderry, Nova Scotia.
Henry A. Laughlin,	Pittsburgh.
Robert G. Leckie,	Sherbrooke, Quebec, Canada.
Charles A. Martine,	Georgetown, Col.
James Neilson,	Youngstown, Ohio.
Ed. M. Parrott,	Greenwood Iron Works, New York.
Ed. C. Potter,	Chicago, Ill.
James B. Randol,	New Almaden, Cal.
Ellen H. Richards,	Boston, Mass.
J. Avery Richards,	Boston, Mass.
Charles Robb,	Montreal, Canada.
G. Collier Robbins,	Eureka, Nev.
William A. Thompson,	Troy, N. Y.
Léon Thonard,	Liége, Belgium.
Samuel T. Williams,	Troy, N. Y.
William E. Williams,	Johnstown, Pa.
Thomas D. Wood,	McKeesport, Pa.

ASSOCIATES.

William A. McIntosh,	Pittsburgh, Pa.
Willis U. Masters,	Cleveland, O.
J. P. Fillebrown,	Lafayette College, Easton, Pa.
John Markle,	Lafayette College, Easton, Pa.
W. I. Whilldin,	Lafayette College, Easton, Pa.

Mr. H. D. Hibbard was changed from associate to member.

The following papers were then read by the Secretary:

An Autographic Transmitting Dynamometer, by William Kent, of Pittsburgh; and A New Air-Compressor, by E. Gybbon Spilsbury, of Philadelphia.

Professor B. Silliman, of New Haven, read a paper on Hydro-Carbon or Water Gas as the Basis for Illuminating-Gas and as an Agent in Metallurgy.

The following papers were then read by title:

The Bloomaries of Northern New York, by Professor T. Egleston, of New York.

Relations of Sulphur in Coal and Coke, and the Atmospheric Oxidation or Weathering of Coal, by Dr. J. P. Kimball, of Lehigh University, Bethlehem, Pa.

The New River Semi-bituminous Coal-fields of West Virginia, by S. F. Morris, of Quinnimont, West Va.

Note on the Use of Bone-Black in Purification of Illuminating-Gas, by Professor B. Silliman, of New Haven, Conn.

Notice was given of a proposed amendment to the Rules, to be acted on at the next annual meeting, to abolish the distinction of home and foreign members.

The Secretary announced that the Council of the Institute had authorized him to act as employment-agent for members and associates of the Institute. Members desiring to obtain employment may send their names to the Secretary, giving the kind of position desired, together with an outline of previous experience. Persons, whether members of the Institute or not, applying to the Secretary for information in regard to suitable persons to fill positions at mines or works, will be given the names on the Secretary's list. It is not intended that the Secretary shall exercise any discretion in the matter, or be involved in any negotiations further than to give to employers the names on his list which correspond to the position which it is desired to fill.

It was also announced that the Council had received an invitation from the Council of the city of Staunton, Va., to hold the next spring meeting of the Institute in that city. The invitation would be acknowledged with thanks and the subject come up for formal consideration at the February meeting.

The following resolution, offered by Dr. Raymond, was unanimously adopted:

Resolved, That the Secretary be requested to convey the thanks of the Institute to the corporations, officers, and individual citizens of Montreal, and the Local Committee, by whose cordial hospitality and wise arrangements the pleasure and profit of this meeting have been secured.

The meeting was then declared adjourned. Immediately after the adjournment the members took a train, courteously provided by the Grand Trunk Railway, to Lachine, returning to Montreal by the Lachine Rapids.

In the evening a subscription banquet took place at the Windsor Hotel.*

On Friday morning excursions were made by train to Victoria Bridge, and by boat to the Lachine Canal and harbor of Montreal.

At 2 o'clock a complimentary lunch was tendered to the Institute by many citizens of Montreal at St. Lawrence Hall. The occasion afforded opportunity for many cordial expressions of sympathy and good-will on the part of the citizens of Montreal and the visiting members of the Institute.

Later in the afternoon, Mrs. Redpath, of Terrace Bank, entertained the members at a garden party at her beautiful house and grounds, thus concluding, most charmingly, the many courtesies shown the Institute during its stay in Montreal.

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* At this banquet there was presented to the Secretary of the Institute a testimonial amounting to over $3000, in recognition of services rendered the Institute in saving its Transactions, when Pardee Hall, of Lafayette College, was burned, at which time his own library and apparatus were lost. The committee in charge of the testimonial have published the proceedings connected with the presentation of this testimonial in a volume, for private distribution.

PAPERS

OF THE

MONTREAL MEETING,

SEPTEMBER, 1879.

RECENT IMPROVEMENTS IN CONCENTRATION AND AMALGAMATION.

BY JOHN A. CHURCH, E.M., PH.D., NEW YORK.

PECK'S MACHINE GOLD PAN.

THE prospector's pan was the first implement used for saving gold, and its action is so effective that it has never been equalled for thorough work. Copper plates, blankets, sluices, and amalgamators of several patented designs have thrust it aside, but in spite of all the advances made there is no dispute among experienced men as to the comparative thoroughness of the work done by it and its rivals. When its rivals are doing their best, the pan can take their discarded refuse and prove their inefficiency by producing from it not only gold, but even the mercury which has been lost from the machines themselves.

The trouble with the pan has always been that it was only a tool. Man was the machine that moved it. His work was high-priced, the quantity treated small, and it was impossible to use the pan for any but the richest sands. As a consequence the best gold-saving appliance known has been discarded for years and replaced by machines of inferior excellence, but which could more than make up for their limited ability in saving gold by their capacity for treating great quantities of ore.

Mr. Peck is one of the old-time amalgamators of the Comstock district, and the first idea of the new machine came to him as he used to pan below the tail-race of his mill. No matter how fine the work of the mill was, the pan invariably proved that better could be done. The tailings always contained gold, silver, and mercury. This state of things is just as true to-day, after all the improvements that have been made, as it was then, and the much-vaunted method of the "Washoe process" consists in extracting 72 per cent. of the gold and silver by one operation, and then repeating this operation half a dozen times to get out about one-third of the remainder. With all the advances of twenty years in milling, there is nothing more crude in metallurgy than the treatment of tailings in many of the large mills of the West.

The fact that the deficiencies of mill work could always be exposed by the pan led Mr. Peck to look upon the improvement of that tool as the true road to the successful treatment of free gold ores and the concentration of all ores by water. His plan was sensible from the start, but it required several years to bring it to successful working conditions.

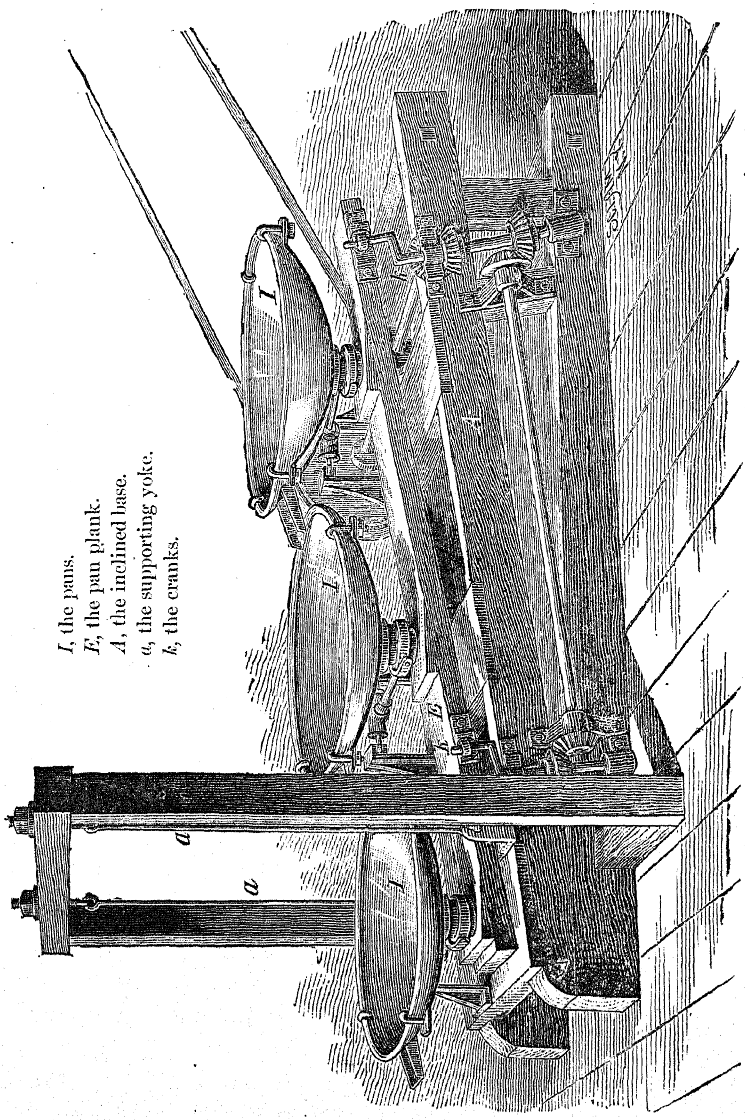
The plan was to raise the pan from the condition of a tool to that of a machine. Ten years ago a Paris engineer invented a "steam workwoman," and Mr. Peck has produced a combination of movements which make his construction in reality a steam prospector.

This was no easy task. The movements of the prospector in handling his pan are so peculiar that the textbooks pronounce them usually to be "indescribable." There is a rotary motion, combined with tipping of the pan, an upward throw, an outward swash of the water, and besides these the rotation of the pan is intermittently interrupted or even reversed. As a result of all these movements the gold and black sand, which are much heavier than the light silicious grains composing gold sands, are collected in the bottom of the pan, and by a repetition of the process these are separated and the gold so completely eliminated from its dross as to be visible on the edge of the black sand.

All but one of the prospector's movements have been produced by simple gearing in Mr. Peck's machine. The exception is the tipping of the pan, which is rendered unnecessary by the immensely increased quantity of materials operated upon and the much greater energy of movement within the large pans employed by him. The means by which this result has been obtained, after so many have failed in attempting it, are unexpectedly simple. A long plank, set at a slight inclination, carries on its surface three pan frames and pans. Its lower end rests on a swinging frame hung from a crosspiece carried on two posts, and its upper end rests on an eccentric of four inches throw. It is by the revolution of this eccentric that the pans receive their upward toss, and it will be observed that this toss varies in amount with the distance of the pan from the eccentric. It is twice as much for the first pan as for the second, while the second and third pans have the same throw, being placed on the plank at equal distances above and below the lower point of support. This is done to make them balance each other and lessen the power required.

The rotation of the pans is accomplished in a simple manner by

two small cranks, which are attached to the plank and by their motion cause every point on it to describe an eight-inch circle. Each



I, the pans.
E, the pan plank.
A, the inclined base.
a, the supporting yoke.
k, the cranks.

PECK'S MACHINE GOLD PAN.

pan is such a point, and each one accordingly is swung around the eight-inch circle precisely as the prospector swings *his* pan around a much smaller circle.

But the prospector's curve of rotation is an interrupted one, or it is an ellipse instead of a circle. He uses both methods with the intention of producing a difference between the motion of the pan and the motion of the water. This was the most difficult part of the human machine to imitate, but Mr. Peck has succeeded admirably. It is a fact of common observation that when a pail of water is quickly revolved the water remains nearly stationary, and when the liquid is in motion if the revolution of the bucket is stopped the water continues to move. The liquid and its containing vessel act independently of each other except to the extent of the moderate friction between them.

It is this difference between the action of liquid and solid under movement that the prospector takes advantage of, and that is reproduced in Mr. Peck's machine. The whole system of pans and the slimes contained in them are thrown into rotation by the cranks spoken of, and a special appliance is provided to arrest the motion of the pans, leaving the slime to continue its swashing rush around the pan.

This appliance consists in an arm which passes through the frame of each pan, and is attached to a crosshead working in fixed guides fastened to the base of the machine. In consequence of the complex movements to be provided for this little crosshead has curved bearing faces, by which it accommodates itself to the circular movement of the plank. To take up the motion of the eccentric the aperture in the pan-frame through which the arm passes is lengthened to a slot, and the arm is held in place by a pin which allows vertical movement. Freedom of rotation is also provided for at the crosshead, the connection with the arm not being a tight one, but sufficiently free to allow the turning of the arm within the crosshead.

It is often the case that a description of mechanical movements is much more intricate than the machinery itself, and that is true in this case. With all the provisions for so many various and peculiar movements, the parts of the apparatus are few and simple. The eccentric, the cranks, and the cog-wheels by which motion is transmitted are certainly common, and the crosshead, which has its bearing surfaces turned to a circular form, can hardly be called a complicated construction.

The pans are usually made forty-two inches in diameter and six inches deep. The plank carries three, standing at successive levels in consequence of the inclination given the plank. Each one carries a spout or nose which projects over the next pan below.

The result of the mechanical combination described is that the metalliferous slime is thrown into a strong circulatory motion by the action of the cranks, while the pans are prevented from sharing this movement by the arm which attaches them to a fixed point outside the movable parts of the machine. But this is not the only action of the arm. The water is thrown into rotation, not by revolving the pans on their centres, but by carrying them bodily around the circumference of an eight-inch circle. Their relation to a fixed point changes at every point of this circle, and this change can be counteracted only by a positive movement of the pan. This is accomplished by the arm and oscillating crosshead, which swings the pan to and fro just eight inches, or the amount of the double throw of the cranks.

This oscillation of the pans has a very important effect in the working of the machine. A good invention will often have happy coincidences which the inventor has not foreseen, but which are really due to the excellent adaptation of the means employed. In this case the spout of each pan, of course, hangs over the *side opposite the spout* of the following pan, and by the ordinary conditions of oscillation or rotation the spout and underlying side of the pan below are moving in opposite directions. The eight-inch oscillation of each pan becomes a sixteen-inch movement of the two relatively to each other.

The oscillation of the pans is in the direction opposite to the throw of the ore pulp, so that when the pulp is rising on the *left* hand side of a pan, the pan above is pouring into it a fresh supply of pulp on the *right* hand side. By this coincidence the fresh supply of slime is not mixed with that which has been under treatment in the pan, but follows around after it to the discharging-point, and accordingly has time for concentration.

Under these circumstances the pulp is thrown directly against the amalgamating surface just as it is partly uncovered by the ebb of the circulating water; and it is thrown *forcibly* against it. The nose of the pan moves in a fifty-inch circle, over an arc sixteen inches long, about sixty times a minute. The pulp is, therefore, dashed against the quicksilver, and the value of such a throw is well known to amalgamators who obtain a considerable part of their gold from plates in the mortar where they can be reached by the gold as it is splashed out by the stamp.

The action of the eccentric is according to the well-known princi-

ple that under a given impulse light particles are thrown further than heavy particles. Quartz sand is projected further than the heavier gold, and the separation effected in this way.

The operation of the machine is as follows: Ore is stamped in an ordinary battery, or crushed between rollers, and either led directly to this amalgamator, or first passed over plates, blankets, or other forms of gold-saving apparatus, and finally brought into the machine pan. In either case it enters the upper pan, and there by the energetic action of the cranks it is thrown around the side of the pan, the strong "swash" keeping the sand from settling or packing. While the ore is kept in this state of diffusion the eccentric acts.

The theory of the prospector's pan is undoubtedly to be derived from the laws of centrifugal force, which has been shown to be the most powerful agent applied in the treatment of ores by water. The light gangue rises to the edge of the pan, and passes through the spout, while the heavier particles gather in the central portion of the pan, where the quicksilver also collects. While in this condition of diffusion the eccentric acts upon the ore precisely like a percussion table actuated by a crank instead of a cam. The heavy particles are thrown toward the "head," which in this case is the bottom of the pan, where they collect, while the light sand passes over the spout. Here it enters the second pan at the lowest ebb of the "swash," and in this pan it is again acted upon by the cam, but with a gentler movement, and a certain quantity of heavy material is separated here. Finally the process is repeated in the third pan.

In practice the second pan collects about one-third and the third pan one-sixth as much rich mineral as the first pan. One of the advantages of the machine is that the lowest or last pan is the best concentrator of the three, so that the poor pulp has the best work of the apparatus applied to it.

The reason for this improvement in efficiency is perhaps partly due to the lessened throw of the eccentric and partly to the existence of a tremor at this point. The two cranks which give the plank its motion around a point are necessarily attached to the plank by yokes, which in consequence of the eccentric's motion have to be reamed out so as to give the crank contact only at the centre line. There is therefore some "lost motion" and a little shake to the plank. The best effect of this shake is obtained at the lowest pan, which receives a real tremor that seems to assist the separation of heavy mineral. Such movements are not unknown in ore-dressing

machines, and the dolly-tub could be operated in this way as well as by the hammer.

It may be observed that the machine pan has been spoken of hitherto only as a concentrator, but that it will be a good amalgamator follows as a necessary consequence of good work as a concentrator. Gold is the heaviest substance treated by the miner, and the uniform practice in amalgamation is to place mercury at the lowest point and then conduct the ore over it in such a form that the gold also can reach this lowest point. When the two metals are in contact amalgamation ensues, and this contact is the absolute prerequisite to success. Even when this contact is provided gold will not always amalgamate, but that is a state of things which no *machinery* can cure, and I am becoming more and more doubtful whether chemical agents will increase the efficiency of mercury as an extractor of such gold from its ores.

The pans are of copper, the interior surface being electroplated with silver and amalgamated, and in the nearly flat bottom a small quantity, four to eight pounds, of fluid mercury is kept. This addition prevents the "drying" of the amalgam, which is kept in a state of saturation by the mercury.

When used for concentration solely, the pans are not amalgamated, but no other change is needed except to work the machine at a slower speed, and cone pulleys provide for the nicest regulation of this speed. All kinds of ore can be treated and concentration carried to any desired extent merely by altering the speed, for this governs the energy of the "swash," which keeps the ore suspended in the water, and finally throws the tailings out through the spout.

The speed for concentrating is lower than for amalgamation, and the limits may be given at forty and eighty revolutions per minute. For concentrating sulphurets the speed is less than for concentrating galena, and this in its turn is lower than for amalgamating, which in effect is merely concentration carried to such a point that all the minerals except gold and mercury will be swashed out of the pan.

When we examine the theory of Mr. Peck's method of amalgamation, we find that it contains an important principle not applied by any other amalgamator. This is the principle of retaining the gold in the amalgamating vessel whether it amalgamates or not.

Regarding the pan as a concentrator merely, and one geared to a speed which will throw out a mineral as heavy as galena, it is plain that gold, mercury, silver, or any alloy or amalgam of these metals, would be retained in the pan.

The gold may be "greasy," or "floury," or "fine," or "light," and refuse to amalgamate on this account, but if it is not in a condition to bid defiance to the laws of gravity it will remain in the pans if the machine is run at the proper speed. The heaviest mineral that the pans will be required to throw out is galena, and the lightest gold which they are likely to treat may be supposed to have a composition of twenty gold and eighty silver. With such conditions the materials in the pan and their respective specific gravities will be as follows:

Specific gravity,	2.6—quartz.	Specific gravity,	10.5—silver.
"	2.6—clay and slate.	"	12.27—gold 20, silver 80.
"	5.0—magnetite.	"	13.57—mercury.
"	5.0—pyrite.	"	19.33—gold.
"	7.7-7.15—galenite.		

From this table it appears that with proper limits in size of the ore, a rate of speed sufficient to throw out galena may be maintained without danger of losing silver or any alloy of silver and gold. In such a state of things no precious metals should go into the tailings except those particles that are attached to grains of sand, or are contained in the pyrite.

Some of these machines have been sent to California, and also to the Snake River country, where valuable auriferous sands are found, but with gold so fine as to defy previous efforts to save it. In general the cheapness and simplicity of the machine are strong recommendations, but the great claim made for it, aside from the attempt to make the favorite prospector's pan available for large operations, is that it includes a real novelty in the theory of amalgamation. This is the principle of retaining the noble metals in the pan whether they amalgamate or not. That method is not used in any other amalgamator, and it seems to me to be a sound and useful principle.

PADDOCK'S PNEUMATIC SEPARATOR.

The Paddock pneumatic ore separator is a jig, in which air is used instead of the fluid more commonly employed—water.

Like the water jig, it consists of an inclined sieve, over which the crushed ore is passed, and through which the air is blown into the ore. The air current is intermittent, and being proportioned in strength to the thickness of the ore bed, the latter is lifted bodily,

and falls again upon the sieve a great number of times during its passage through the machine. By this treatment it is separated in layers according to the specific gravity of its constituents, the heavier minerals being collected on the sieve with the lighter over them. This separation is not difficult, but it has been found on trial that in this simple form the machine is by no means perfect. Mr. Paddock, who is a miner of copper ores, built a concentrating mill, where he gradually devised one improvement after another to meet the difficulties encountered.

Where there is a change in the size of the sand, or in the minerals which an ore contains, or even in the proportion of the minerals which compose it, the time during which the ore remains under treatment must be altered, or there is danger of mixing the separated minerals again.

Mr. Paddock provides three modes of preventing this remixture. One permits the shortening or lengthening of the ore bed, another changes the inclination of the sieve on which the ore rests, and the third gives a horizontal movement in opposite directions to the rich and poor mineral.

1. *The Adjustable Hopper.*—The ore is delivered upon the sieve through a hopper, and flows down to the tailboard with as much steadiness as if it were fluid, the separator being continuous in its action. As it flows down fresh supplies run in from the hopper, and it is evident that the position of the point of supply governs the length of the bed and the time of treatment. Mr. Paddock therefore makes the hopper adjustable, so that it can be moved down from the extreme head about one-third the length of the machine.

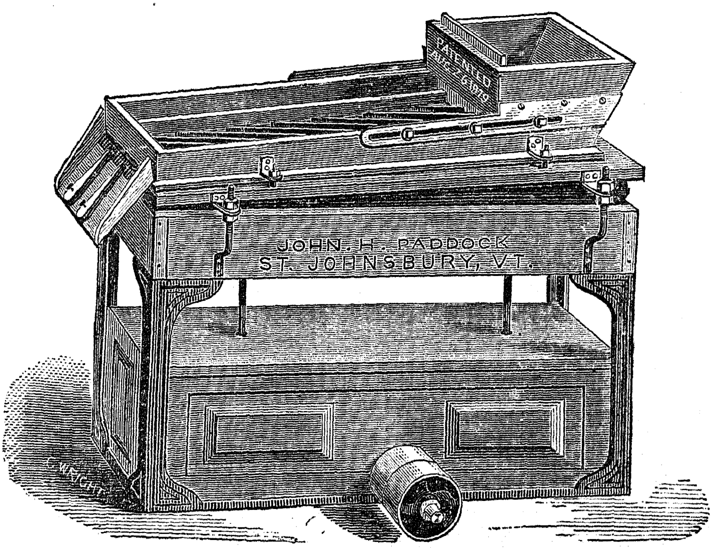
2. *Inclination of the Bed.*—The air-chamber under the ore bed is made with flexible sides, so that its inclination can be altered without disturbing the motive parts of the machine. This change can be made either lengthwise, by which the rate of flow of the ore is increased or diminished, or sidewise, in which case the discharge of the separated minerals is controlled in favor of the rich or poor mineral. The first of these alterations evidently produces results of the same kind as the adjustment of the hopper. The operation of the second will be understood from the following paragraph.

3. *The Discharge Grating.*—The final discharge of the ore is over a tailboard at the end of the machine, where heads, middlings, and tailings run off side by side, instead of one over the other as in wet jigs, and to produce the horizontal movement necessary to place them

side by side an inclined grating is laid on the cloth which in air jigs forms the sieve. This grating is made in two parts. The lower one, made of narrow strips of sheet brass inclined to the path of the ore, directs the concentrated mineral to one side of the bed, where it collects in a trough and runs down to the discharge without danger of mixing again with the gangue. Upon these strips there is a similar grating made of broader strips, which are inclined in the contrary direction.

In its progress down the bed the ore is separated into two layers, according to the specific gravity of its constituents, and each of these layers is led to its own side of the machine, forming the heads and tailings. Between them at the tailboard a mixture of the two is obtained in small quantity, and this composes the middlings. The whole bed is slightly inclined toward one of the lower corners, that on the side of the heads, and the flexible sides of the air-chamber permit this inclination to be altered until just that slope is formed which gives the best separation. All these changes can be made without stopping the work, and in a moment.

All these modes of adjustment are shown in the accompanying



figure, but the elastic diaphragm, which answers the purpose of a bellows, and the motive parts are both concealed by their coverings. The frame is mostly made of wood, the corner standards only being of iron.

If we examine the system of wet jigs in Europe we shall be struck by the fact that they seem to have been made to satisfy previous calculations of what would be needed, but a very different impression will be obtained from American works. In Europe there is little adjustability, except in speed; in America the favorite machines permit the most complete adaptation to the work which is momentarily in hand, and this without a moment's stoppage. Something of the same kind seems to have been needed in dry jigging, and Mr. Paddock has supplied it.

The number and force of the air puffs are also varied at will. The "action" of the machine is placed on the floor in a tight box, where it is kept from dust. It consists of cams that operate lever arms, the length of which can be altered at will. Air has been used for many years as a means of separating heavy and light, or coarse and fine minerals, but the older efforts seem to have been confined to attempts to winnow the ore as grain is winnowed. The finely crushed ore was blown into a long chamber with the expectation that the light particles would be carried farther than the heavy, but after repeated trials this system has received general condemnation.

Mr. Thomas J. Chubb, in attempting some years ago to devise a method for treating the gold sands of California, made the first air jig, and his ideas have been the source of all the systems of air concentration which now exist in America. He went to California with a machine, but died there before he could procure its adoption in the mines.

Mr. Paddock purchased Mr. Chubb's rights from the administrator of the estate, and after great trouble succeeded in finding the original Chubb machine in San Francisco. It had "good ideas," as Heine's tailor said of the old overcoat which the poet took him to be made into a bed-gown, but it proved to be deficient in several necessary points. The whole machine was carefully studied and improved by Mr. Paddock, who is an experienced mechanician, and the result is an air jig, which is,

1st. Adjustable to different rates of work with the same ore.

2d. Adjustable to different kinds and sizes of ore.

3d. Extremely rapid in action and thorough in work.

In presenting this system of concentrating ores to the notice of the Institute, it is almost necessary to consider the objections which have been made to the use of air as a means of separation.

One is that the ore must be dried artificially. In practice this has not been found to be true, but I cannot say that ores from un-

usually wet mines have been worked. The machines are in constant use, and no means are employed for drying the ore, which is worked just as it comes from the mine, and the mines at which the machine has been used demand the usual means of drainage.

Another is the production of dust. It is a cardinal principle of wet-ore dressing that the ore shall not be broken finer than is absolutely necessary to separate the rich mineral from the lean rock. This lessens the production of dust, which in treatment with water is always liable to float off. It is now not uncommon to find water jigs that treat ore of $\frac{3}{4}$ and 1 inch diameter.

But this precaution is impossible in air jigs, for the ore must be fine to be moved by the air puffs. Mr. Paddock has usually treated ore that had been crushed fine enough to pass through a 40 or a 60 mesh, but this fineness is not absolutely required by the machine, which has treated $\frac{1}{16}$-inch material, and it is thought can treat ore of $\frac{1}{12}$-inch size. It has also treated with success ore which passed through a mesh of 100 to the linear inch.

With ore of 40 and 60 size the amount of dust is about 15 to 17 per cent. This goes on the bed with the coarser ore, and being blown out by the air is drawn by a suction fan and thrown to a cloth-chamber, where the air is discharged and the powder collects. This powder is then washed upon a slime machine, and Mr. Paddock prefers that of Mr. Hooper, the able manager of the concentration at the Ticonderoga Graphite Works.

There is no doubt that the crushing for dry concentration produces more dust than the crushing for wet jigs, provided the work is done by rolls in both cases. But the amount of dust is not very great, and there are advantages in air jigs which must go far in any situation to counteract the disadvantage of the dust. These advantages are cheapness, quantity of product, and freedom from loss by float mineral. A plant suited to the treatment of 30 to 50 tons of ore daily can be built in the Eastern States, with all its breakers, rolls, sieves, and jigs, for about $10,000. One machine will treat from $1\frac{1}{3}$ to $1\frac{1}{2}$ tons per hour, and three machines will treat 50 tons daily, and work up the middlings. The present cost of treatment is reported at 50 cents per ton, the work being done on the scale of more than 30 tons daily, and at such a cost it is evident that extremely poor ores can be made available.

This is the true test of mining. It is well known that the costs of work are greatly increased when the ground is "pockety," and all mines become pockety when none but ores of the highest grades

can be taken out and treated. The conditions for safe and profitable mining almost necessarily include the economical removal and treatment of the middle grade ores. Frequently this can be done only by joining some system of concentration to the mine, and all countries contain innumerable situations where this can be done best, or done only by means of air.

The advantages of dry concentration do not depend upon a careful balance of what it will do in comparison with the wet treatment. There are great numbers of mines, both West and East, where water cannot be had in quantity, except at considerable and constant expense. Many mines are lying idle, or only worked for their richest ore, because no means of concentration, successful in itself, and adapted to their conditions, has been presented. In such a case there is no need of urging the claims of any system of concentration to be acknowledged the best in the world. The problem is to produce one that will work profitably in that situation. Wherever water sufficient for making steam can be had these machines will do good work on ores of iron, lead, and copper containing gold and silver, or not. I do not go further and place gold sands and other ores of the precious metals in the list, simply because the Separator has not yet been tried on them, but there is no reason known for excepting them from the list of minerals that can be profitably treated by this system.

NOTE.—When this paper was read before the Institute I was asked by Professor Monroe for comparative analyses of the ore and tailings, but was unable to give them. Since then the results obtained at Blue Hill, Maine, where six Paddock separators are working on ores of copper and lead containing silver, have been supplied me. Mr. C. W. Kempton, who first brought this district to the notice of the Institute, wrote October 1st, 1879, as follows: "The dry machines separated the ore (galena and copper, with some iron) from the rock most completely, giving tailings assaying from $\frac{7}{10}$ oz. per ton up to $1\frac{1}{4}$ oz. [Probably this refers to silver in the tailings.] I am of the opinion that nothing would prevent working even closer with sizing arranged especially for silver ore. We worked on ore which assayed only 4 ounces per ton with very profitable results, and found no more loss in higher grades. The No. 3 size machine took from 60 mesh to 100 mesh, and separated finely, but should use an intermediate machine, say 80 to 100, for steady silver work. After separating the rock from the ore we ran the ore through the dry machines a second time, separating the galena from the copper and

iron most completely." In working the slimes (passing through 100 mesh) the loss in tailings was found to average 2 ounces per ton, and Mr. Kempton thinks this can be reduced.

These results are sufficiently favorable to indicate that the important problem of thorough ore concentration by air has been solved. The cost of treatment at Blue Hill was reported by a New York *Tribune* correspondent at 50 cents per ton. The field for this method of concentration is not confined to that vast region in the West, where water is so scarce as to be sold for 10 and 20 cents a gallon. Even in New England there are many places where the conveyance of water to the mines in sufficient quantity would cost more than the mill machinery, and in such localities the engineer has hitherto been confined to a balance between the cost of transportation to the nearest water and the cost of an extensive water service. The introduction of dry concentration will permit the erection of the works at the mine's mouth, where the ore will be delivered to the mill without adding any cost to the mining expenses.

DISCUSSION.

DR. R. W. RAYMOND: The idea of imitating the movements of the prospector's pan, in an ore-separating machine, has been a favorite one with inventors; but I think it is based on two errors, the first being an exaggerated notion of the delicacy of the pan as a means of separation, and the second an overlooking of the fact that the motion given to the pan is perpetually varied by the operator, according to the dictates of his experienced eye. No man ever could wash a panful of "gold dirt" with his eyes shut. The efficiency of the pan, such as it is, turns upon its controllability by the hand and eye. It is like a jack-knife in carpentry, useless without the skilled operator. A Yankee can make anything with a jack-knife; but a machine to imitate the movements of the knife, omitting the Yankee, would not whittle with success. Moreover, the pan, with a man attached, is still a very imperfect affair. It is less accurate and delicate than good machines for anything like rapid work. The operation of panning is close because it is slow, and not merely because it is peculiarly adapted to the separation of particles. It is employed without sizing, and the operator pauses from time to time to remove with his hand the larger pebbles and much of the apparently worthless material, which he closely scrutinizes before throwing it away. I do not think these manipulations can well be

imitated mechanically; and until we have actual working results from the machine-pan Professor Church has described I must doubt its efficiency.

MR. MACFARLANE: I fully agree with the remarks which have been made by Dr. Raymond. The gold-pan of the prospector is much less capable of imitation on a large scale than the hand "Sichertrog" in almost daily use by the German *Pochwerkssteiger*. The latter instrument has given rise to the percussion table, and distantly to Von Rittinger's continuously-working table with side stroke, and yet these machines have not been adopted in dressing-works in this country. I doubt whether in this case or any other an amalgamator and concentrator could be advantageously combined. At certain gold mines in Marmora, Ontario, amalgamation in the battery or on plates has been abandoned in favor of concentration on a Frue vanner, calcination of the concentrate and subsequent amalgamation in pans. In this manner 75 per cent. of the gold contents of the vein-stone have been recovered at the Dean Gold Mine, the ore of which consists mainly of arsenical pyrites.

DR. CHURCH: I think that Dr. Raymond's criticisms of the machine-pan are not well founded. It does not size the ore simply because it is not constructed to size it. Its function is to take properly prepared pulp and treat it in the most effective way. It is peculiarly adapted to the separation of particles, for the reason that the rotary movement is the best and most delicate means known for the treatment of ores. The human supervision of which he speaks cannot be estimated at a very high value, for it is altogether a blind and ignorant control of the work. The scrutiny of a man who cannot see what he is doing, and could not understand it if he did, cannot be of much service. Then, too, in regard to Mr. Macfarlane's objection that the machine-pan cannot possibly take out all the gold, because some of it is so locked up in sulphurets that no mechanical work will liberate it—that is certainly true, but it is no criticism of this machine, which undertakes to do only that which can be done by mechanical means. To obtain the gold from these sulphurets a means of concentration must be used, and this means the pan supplies. It is a concentrator, and one which applies the principles of concentration to amalgamation. It embodies a distinct and novel principle in amalgamation,—that of retaining all the gold and silver whether they will amalgamate or not, and I believe this principle to be a sound and valuable one.

WASHING PHOSPHORIC PIG IRON FOR THE OPEN-HEARTH AND PUDDLING PROCESSES AT KRUPP'S WORKS, ESSEN.

BY A. L. HOLLEY, C.E., LL.D., NEW YORK CITY.

THIS process is performed in the Pernot puddling furnace; it removes from 75 to 80 per cent. of the phosphorus, most of the sulphur, and practically all the silicon, from crude iron, in from five to eight minutes. It has been in regular use since March, 1877, and has produced over *seventeen thousand tons* of washed metal for the open-hearth furnace. During two years, nearly all, and lately, all the open-hearth steel produced in these works has been made from about 5 tons of washed pig and 2 tons of scrap per open-hearth furnace heat. There are 12 open-hearth furnaces, of which 4 to 9 are running, according to the state of orders. A large amount of highly phosphoric iron from France and Belgium has been also washed and then puddled for parties who have taken, or are expected to take, licenses. The washing of pig for puddling, however, is not a practice at Krupp's works, because pure pigs are nearly as cheap as impure ones.

The author and his assistant, Mr. Laureau, spent, during the last month, three days at Krupp's works, examining this process. They copied out of the office books characteristic analyses and physical tests, some of which are given herewith. They also brought home many samples of pig, cleaned metal, puddle bar, and steel, which they saw treated. Analyses of some of these samples appear in Table IV.

As there are no blast furnaces at Essen, the iron is melted in two Bessemer cupolas with fore-hearths; they melt 12½ pounds of iron with 1 pound of coke. The metal is run by a spout from the fore-hearth into an opening at the side of the main door of the washing-furnace.

The Washing-furnace.—This is a regenerative gas furnace. It has a Pernot revolving-hearth of 12 feet external diameter and 3 feet depth. (See Plate.) The four regenerator chambers have 780 cubic feet capacity, which is about the same as the average regenerator capacity of open-hearth steel furnaces of equal tonnage. The lining is 13 inches thick on the sides and 9 inches on the bottom, thus giving a hearth 9 feet 10 inches by 2 feet 3 inches deep. The lining is composed of lumps of highly refractory ores

roughly fitted together, the interstices being filled with ore, and the whole being glazed at a melting temperature. Large lumps are placed on the sides and smaller lumps on the bottom. When the fine ore has melted and run between the lumps, more fine ore is put on and melted, until the lining becomes monolithic. The hearth is then fettled.

The Fettling.—This averages 20 per cent. on the pig-iron charge, but more is used with irons very high in phosphorus. Iron ore alone has been used; also hammer scale alone; usually ore with a little hammer scale is employed. The charges run out very clean and hot. After each heat, the gas is turned off for five minutes while the tap-hole is turned on the high side and redressed. The bottom sides, or lower part of the slopes of the hearth, which have been most eaten out during the process, are then filled with fettling-ore wet with just enough water to make it stick together, so that it can be readily handled, and so that it will not blow over into the regenerators. A long-handled, large, shallow spoon is placed across a bar in the charging-door. One workman shovels the fettling into the spoon; another throws it out of the spoon against the slope. After each two or three spoonfuls, the hearth is revolved a little, so that the fettling is always conveniently dropped in the same place relatively to the door. This operation occupies sixteen to twenty minutes.

The fettling should contain a minimum of 6 per cent. silica, with a maximum of 15 per cent. If the silica exceeds 15 per cent., or if the silicon in the pig exceeds 1 per cent., it is best to add as much lime as there is silica in the ore (a little lime is always useful); if the silica is less than 6 per cent., the fettling will not adhere.

The maximum temperature, which is above high-puddling heat, but considerably lower than open-hearth steel heat, is kept up between, as well as during the operations; this temperature slightly melts the surface of the fettling, and sometimes melts furrows 2 or 3 inches deep in the less refractory parts. During this time, the hearth is revolved 3 or 4 turns per minute.

The Irons Used.—The charge is from 5 to 7 tons—usually 5 tons. Messrs. Bender and Narjes, who have developed the process, insist that at least 0.30 per cent. of manganese is essential to the most economical result, even if it has to be added in the shape of spiegel-eisen. They prefer 1 per cent. manganese. It seems quite certain from analysis that manganese protects carbon from oxidation, and so keeps the bath very hot and fluid until the phosphorus is re-

moved. Carbon should also be as high as possible; at least 2.7 per cent. As little as 2.5 per cent. has been employed, but 3 per cent. is preferred.

Silicon should be as low as possible. If it is higher than 1 per cent., lime must be added; there is a greater waste of fettling, and the operation is prolonged. The silica in the slag must be less than 20 per cent.; if it runs from 20 to 30 per cent., only 20 to 30 per cent. of phosphorus can be got out of the iron.

For steel, the pig iron used averages 0.70 to 0.80 in phosphorus; this element, after washing, is reduced to 0.10 to 0.15, and may be still further diluted by pure scrap. Silicon and manganese are reduced to traces. Table I gives an average result. A large amount of *Flusseisen* is made containing phosphorus 0.15 to 0.20; but the carbon is about 0.08 and the silicon very low. This metal has an elastic limit of 15 to 18 tons, an ultimate tenacity of 26 to 30 tons, and an elongation of 23 to 29 per cent.

TABLE I.—PHŒNIX PIG, CLEANED FOR OPEN-HEARTH STEEL.

	C.	Si.	P.	S.	Mn.	Cu.
Raw pig	3.30	0.39	0.74	0 09	2.32	0.14
4 min. washing	3 27	0.02	0.16	0.024	0.038	0.15
5½ min. "	3.27	0 01	0.146	0.026	0.116	0.14
7 min. "	3.32	0.023	0.106	0.029	0.058	0 143

	Silica.	Ox. of Iron.	Ox. of Mn.	Alumina.	Lime.	Phos. acid.	Sul. acid.	Copper.
Slag....	13.0	51.0	16.6	11.6	0.7	6.0	0.2	trace.

Open-hearth charge. { Cleaned pig .... 11,000 lbs.; Steel turnings .... 1,650 "; 50 per cent. ferromanganese... 198 " } Total, 12,848 lbs.

	C.	Si.	P.	S.	Mn.	Cu.
Steel plate	0.10	0.025	0.081	0.03	0.194	0.11

STEEL PLATE:

Tenacity, lengthwise	Tons, 25.87
Tenacity, crosswise	Tons, 26.20
Elongation, lengthwise	Per cent., 29 to 31½
Elongation, crosswise	Per cent., 25½

TABLE II.—ILSEDE PIG, CLEANED WITH ILSEDE ORE, FOR PUDDLING.

	Silica.	Ox. of Iron.	Ox. of Mn.	Alumina.	Lime.	Magnesia.	Phos. acid.	Carb. acid.	Water
Cleaning ore.	4.80	52.60	8.36	1.20	11.73	0.80	3.83	7.90	7.88
Slag............	10.40	41.00	19.30	2.00	7.30	0.70	20.00*	......	......

* Limit of saturation, 23 per cent.

	C.	Si.	P.	Mn.
Raw pig..	2.50	{0.20 0.31}	2.92	2.61
Cleaned pig.....	2.40	traces.	{0.65 0.90}	traces.
Puddle bar.............	......	....	0.088	....

TABLE III.—LUXEMBURG PIG, CLEANED FOR PUDDLING.

	Silica.	Ox. of Iron.	Ox. of Mn.	Alumina.	Lime.	Magnesia.	Phos. acid.	Sulph. acid.	Water.
Cleaning ore.	7.6	65.0	0.2	3.0	10.2	2.7	2.2	0.1	9.0
Slag............	9.5	70.3	2.1	4.1	1.9	0.3	11.5*	0.3	.....

* Limit of saturation, 16 per cent.

	C.	Si.	P.	Mn.	S.
Raw pig..	{3.40 3.10}	0.46 0.59	2.23 2.09	0.18 0.19	0.16 0.38
Cleaned pig..	.....	trace.	{0.65 0.95}	trace.	......
Puddle bar...	......	....	{0.06 0.15}	none.	......

For puddling, the highest known amount of phosphorus in the pig, say 3 per cent., may be washed out to 0.65 to 0.90, and this in puddling is reduced to 0.06 to 0.15 per cent. The United States Test Board's experiments on chain-cable irons show that much more phosphorus than this, if carbon is low, may exist in the toughest irons. Tables II and III give average results.

The Washing Process.—The crude iron charge from the cupola is from 5 to 7 tons. It begins at once to bubble in the Pernot hearth, from the mechanical action of pouring. As soon as it is all in, the revolutions of the hearth are increased to 11 per minute. Large blotches of slag soon appear with iron spouting through ; but this is no criterion of the state of the bath. Usually in 2½ or 3 minutes from the time the iron begins to run into the furnace, the bath rises and the slag flows more or less out of the joint between the revolv-

TABLE IV.—TREATMENT AND ANALYSES OF PIG WASHED AND PUDDLED AT KRUPP'S WORKS, AS OBSERVED BY THE AUTHOR, AUGUST, 1879.

No. of the charges.	Date, August, 1879.	Cupola tapped at	1st ball withdrawn at	Weight charged.	Weight of bars.	DESCRIPTION.	ANALYSES. Original. C.	Original. P.	Original. S.	Original. Si.	Original. Mn.	Cleaned Pig. C.	Cleaned Pig. P.	Cleaned Pig. S.	Puddle Bars C.	Puddle Bars P.	Puddle Bars S.
1	12th	8 h. 15 m.	8 h. 45 m.	1,100 lbs.	1,025 lbs.	80 per ct. Metz, and 20 per ct. Phœnix.											
2	"	9 10	9 42	1,100	979	" " " "											
3	"	10 08	10 45	1,100	1,042	" " " "	2.90	2.22	0.350	0.31	0.07	3.10	0.42		0.08	0.154	0.031
4	"	11 05	11 40	1,100	1,067	" " " "	3.60	0.65	0.050	0.40	3.00		0.58				
5	"	12 06	12 47	1,100	979	" " " "											
6	"	1 08	1 47	1,100	1,034	" " " "											
7	"	2 02	2 35	1,100	1,012	Metz & Co.,											
8	"	3 03	3 42	1,100	1,034	" "	2.90	1.95	0.334	0.60	0.18	2.58	0.55		0.07	0.140	0.031
9	"	4 02	4 37	1,100	1,078	" "	3.10	2.05	0.389	0.59							
10	"	4 58	5 37	1,100	1,111	Sclessin,											
11	"	5 55	6 40	1,100	1,034	"	2.85	1.65	0.500	0.62	0.17	2.65	0.86	0.15	0.06	0.114	0.031
12	"	7	7 42	1,100	1,012	"											
1	13th	7 45	8 20	1,100	1,012	Metz & Co.,											
2	"	8 40	9 20	1,100	968	" "	2.90	1.95	0.334	0.60	0.18	2.58	0.55		0.08	0.088	0.024
3	"	9 41	10 24	1,100	1,067	" "	3.10	2.05	0.389	0.59							
4	"	10 42	11 30	1,100	1,045	Ilsede,											
5	"	11 55	12 52	1,100	1,078	"	2.50	2.92		0.55	2.61		0.69		0.11	0.088	0.015
6	"	1 20	2 05	1,100	902	"											
				19,800 lbs.	18,479 lbs.												

Coal consumption = 4840 pounds, or 600 pounds per ton.

ing hearth and the roof. If the slag rises earlier than 2 minutes the speed of rotation is decreased. The rising of the bath somewhat represses the bubbling. The rising lasts about two minutes; and after it has fallen a little, the carbonic oxide, with its characteristic flame, begins to blow out, the bubbles on the surface of the bath increase largely in diameter, up to 6 or 8 inches, and are broken here and there by iron spouting through; the ebullition seems a little more sluggish than at the first stage of the operation, but the bubbles are much larger, and increase more and more in size and number as the operation advances. The spouting iron, toward the last, rises from 6 to 10 inches above the bath, and presents the appearance of a miniature forest of trees. The bubbling is not, however, as lively as that of the pig and ore bath, because the temperature is lower. The close of the operation is indicated by a rather sudden and voluminous generation of carbonic oxide and of spouting due to its release. The tap-hole is then brought around to the spout, and the furnace is tapped as soon as possible. Flame constantly blows out of the openings in the furnace from three causes,—the slight gas plenum, the rapid revolution of the hearth, and the generation of carbonic oxide.

The time of the washing operations witnessed averaged between five and eight minutes. Rich fettling, of course, shortens the time. The tapping, from stopping the rotation of the hearth to opening the tap, averaged two minutes, and the time from stopping the hearth to filling the ladle for the open-hearth, or the pig-bed for puddling, averaged 5 minutes.

Running Out.—The washed iron invariably runs out much hotter than it went in; no iron nor slag remains in the furnace. The ladle for one line of open-hearth furnaces stands in a pit about 20 feet from the tap-hole; the other ladle is about 50 feet away. Some slag runs out with the last of the metal; this runs over the ladle into a spout, which conducts it to a slag-pit in the floor. When the metal is all out a section of the furnace-spout is moved laterally, to run the bulk of the slag into a pit. One ladle sits in a car which is raised to the general level by a hydraulic lift and is then drawn to the front of the open-hearth furnace; the ladle is tapped into a spout 12 feet long leading into the furnace door. The other ladle is raised out of its pit by a locomotive crane, which also transports it to the open-hearth furnace. The metal may be held in the ladle twenty minutes without perceptible chilling.

The pig-bed for receiving washed iron for puddling begins about

20 feet from the furnace tap-hole. The slag that runs out with the metal is partly stopped by a skimmer and partly run off the end of the sow. When the metal is all out, a section of the spout is moved laterally to run the slag into a pit, or the furnace tap-hole may be moved laterally to run it on the floor.

These arrangements are obviously not so convenient as they could be made in a new plant. The washed iron should also be kept fluid for puddling as well as for the open-hearth. In these works it is remelted in a cupola, as the puddling plant is a long way off.

Output of the Refining-furnace.—The operation and the repairing and heating (three-quarters of an hour) of the new fettling occupy altogether, on an average, one hour and a quarter, so that nine 5 to 7-ton heats may be made per turn. The regular output is 80 to 90 tons per 24 hours, and the regular wages are based on 80 tons; a large bonus is divided among the workmen on an excess of product. It is obvious that one washing furnace could keep one very large or two medium blast furnaces going.

Waste.—As the impurities of the iron are removed by the oxygen of the ore, there is no waste of iron except 1½ to 2 per cent., which is mechanically entangled in the slag, and is thus wasted, unless the slag is cleaned. The ore comes out as slag; it is too highly phosphorized for use in the iron manufacture, but is worth about forty cents per ton to silver and lead smelters.

The Open-hearth Process.—The open-hearth charge usually consists of 5 to 5½ tons of washed pig and 2 to 2½ tons of scrap, not preheated. With this charge a half ton of ore is used in the open-hearth furnace; but with a 7 to 8-ton charge of washed pig, without scrap, one ton of ore would be used. Some 8-ton heats with 3 tons of scrap were observed. The time of making a 7-ton open-hearth heat as observed, averaged seven hours, and the time of repairing the furnace between heats was about one hour. The increased output by using washed pig in the proportion mentioned, as compared with the old pig and scrap process, is about 1 ton per furnace per shift. The waste of iron is also considerably decreased by reason of the low silicon. The principal products are tires, axles, plates, and forgings.*

Cost.—At Krupp's works the washing process costs at present about as follows per ton of open-hearth steel: Mixture of irons, $12; cupola melting, 85 cents; washing, including fettling, repairs, fuel, labor, etc.,

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* The author saw, nearly completed, a new ram or stem-post for the iron-clad König Wilhelm. It was a complex forging from open-hearth *Flusseisen*, containing carbon 0.08.

$1.25 = say $14. Bessemer pig, however, costs but about $15.25, so that only $1.25 profit is realized. Members can readily apply these figures to different conditions in various parts of the United States. Cupola melting is, of course, unnecessary if blast furnaces are near.

The men employed at the washing furnace are one melter, one helper, and two or three laborers, who wheel ore and slag; also pig-bed men if blast-furnace metal is not used direct. The steel-furnace laborers get the metal from the washer to the steel furnace.

The washing process is without value to the Bessemer manufacture, which depends upon silicon for its converting heat. Repeated experiments in blowing washed metal in a gas-heated converter have been unsuccessful.

Repairs of Washing-furnace.—The hearth is generally, but not always, pulled out Saturday night to repair the lining by means of lumps of refractory ores. The lining is usually kept good by the daily fettling. The roof lasts from six to nine months, excepting the parts around the ports, which last from five to seven weeks. The regenerators must be cleaned every two months.

Experiments at St. Chamond.—The author learned at these works, as well as at Essen, the results of experiments (July, 1879), upon which a Krupp license has been taken by the St. Chamond Company. About a dozen charges of Moselle pig, containing phosphorus 1.90, were washed in a Pernot steel furnace, with ore lining, and yielded phosphorus 0.39 to 0.40. Charges of cold washed pigs 60 to 65 per cent., and hot scrap 40 to 35 per cent. were made into steel in four hours in a Pernot furnace. Messrs. Narjes & Bender, therefore, confidently predict an output of 80 to 90 tons of ingots per twenty-four hours from washed pig in such Pernot furnaces as are erecting in the United States.

Puddling.—The washed pig should be run direct from the washer to the puddling furnace. The arrangements at Krupp's works, however, require that it should be remelted in a cupola. The reason of remelting in a cupola rather than in a puddling-furnace will be understood by iron makers. It is difficult to keep the washed metal fluid during gradual decarburization. The metal first melted in the puddling furnace comes to nature, and incloses some crude metal, thus preventing its conversion. This might be remedied, perhaps, by more mechanical agitation; but in Krupp's practice, premelted metal yields 0.05 to 0.10 phosphorus in the puddle bar, against 0.15 to 0.50 phosphorus from metal melted in the puddling furnace. The temporary cupola used has two feet internal diameter, and its product

runs direct into the puddling furnace through a movable spout; it runs upon a peel held in at the opposite door, so as not to cut the furnace bottom.

An ordinary double puddling furnace, with a Bicheroux gas apparatus, is fettled in the ordinary way with iron ore, hammer-scale, and a little manganese ore; but the manganese ore is sometimes omitted; then a charge of one-half ton is run in. After fifteen minutes' rabbling, the iron began to come to nature, in the heats observed. In from twenty-nine to thirty-one minutes the first balls were drawn. The ninth ball was drawn about eight minutes after the first. The balls were hammered and rolled to puddle bars 4 by $\frac{3}{4}$ inches, which were quite smooth and clean. The washed puddle bars from pig containing from 2.50 to 3 per cent. of phosphorus, were as fibrous, silky, and tough as best ordinary puddle bars. The second charge came out sixty-two minutes after the first. The author kept track of eleven charges made in this furnace between eight o'clock A.M. and five P.M. The fettling averages about 600 pounds per ton, and the loss on cleaned iron with this small fettling is about eight per cent., including cupola loss. The wear of fettling is very small, by reason of the very small amount of silica in the bath. This fact would give the mechanical puddler a great advantage with washed iron.

The economy in fuel is also important; for example, at Dillingen, the ordinary practice with Strumm pig is fifteen charges of 660 pounds each, with 1800 pounds of coal per furnace per 24 hours. The same pig cleaned yielded 20 charges of the same weight with 1188 pounds of coal, or 66 per cent. of that used with raw pig. The quality of product was the same as that of $\frac{1}{3}$ Strumm and $\frac{2}{3}$ Nassau, the latter being of high quality.

Conclusion.—The proportion of phosphorus eliminated by this process is not so great as by the Thomas and Gilchrist process; the efficiency, however, of lime linings and additions in the open-hearth is yet to be tested. The Krupp washing has little or no value in connection with the Bessemer process. The Siemens direct process also yields a material adapted to the open-hearth, more free, not only from phosphorus, but from carbon, than the Krupp washed metal, and in better condition to be converted rapidly into steel. The object of this paper is, however, not to compare refining processes, but to give the complete facts about the Krupp process; and it must be admitted that this process is cheap, uniform, convenient, and thoroughly effective within certain limits, and that it is no longer in any sense experimental.

NOTE ON THE ZINC DEPOSITS OF SOUTHERN MISSOURI.

BY ROSSITER W. RAYMOND, PH.D., NEW YORK CITY.

THE lead-mining industry of Missouri, as of other parts of the Mississippi basin, appears to have been paralyzed by the shock of competition with the mines of the States and Territories further west. It is difficult for establishments working galena ores, too poor in silver to pay for extracting it, to compete with the producers of the argentiferous carbonates of Utah and Colorado, who look upon their lead as a by-product of little value, and are satisfied if it pays the charges of transportation, or the loss in smelting for silver. But simultaneously with this depression in the lead-mining of the Mississippi States, the development of new zinc deposits has caused a notable revival in the mining of zinc ores. There has recently been a sudden and strong rise in the price of spelter, which, though probably to a large extent temporary in character, has given important encouragement to prospectors and miners. Apart from the general recovery of business in the United States from the prostration of the past six years, the cause of the advance in spelter has been the falling off in production, resulting from the shutting down of several leading Western works,—the Martindale and Carondelet works, at St. Louis, and the Consolidated Company in Kansas. At the same time, the principal works at La Salle, Ill., were undergoing repairs, and produced less than usual, while the sheet-mill of Matthieson & Hegeler, at that place, was running as usual; so that this concern, ordinarily a large producer, entered the already depleted market as a buyer. These causes are in their nature transitory, and it is quite likely that spelter will be superabundant in a few months.

Having lately received from a well-informed Missouri correspondent some samples of the zinc ores and a general account of the ore deposits of Southwest Missouri, I have thought that it might interest the Institute to examine these specimens, and to hear such brief statements concerning the region as I am able, upon good authority, though without the aid of any personal investigation on the spot, to give.

The zinc-bearing territory in Southwest Missouri appears to be very extensive; that is to say, zinc ores have been mined during the last year at various points throughout a district extending from the eastern edge of Kansas, west of Joplin, to a line some 50 miles east

of Springfield. Many of these openings, however, have already been abandoned. The deposits were superficial and limited, or the quality was unsatisfactory, or transportation was too dear, etc. The ores were carbonate and hydrous silicate of zinc. Further north and northeast, discoveries of zinc ore are also reported, but no developments have yet been made. These northern deposits, like those of Southeast Missouri, are said to be in Silurian rocks, and therefore belong to a different horizon from those of the Southwest, which are pronounced sub-Carboniferous. I need hardly remark, however, that this difference in age of the inclosing rocks does not necessarily prove a similar difference in age of the deposits, since all that can be predicated upon a determination of the period to which the country rock belongs is the limit, back of which the beginning of the deposit is not to be placed. In other words, an ore-deposit in Silurian rocks must be Silurian or post-Silurian in age, and its formation may have taken place much more recently than that of the country rock. The sub-Carboniferous and the Silurian rocks of Missouri may, therefore, be traversed by deposits of some one age posterior to both.

The sub-Carboniferous limestone of the Southwest lies nearly horizontal, and is characterized by flinty segregations, sometimes solid, sometimes brittle, and much fissured. The limestone is occasionally shaly. The ore-deposits, irregular in shape and distribution, occur in this formation. Those which have proved most productive hitherto have been associated with lead ores; but their superior productiveness is probably due to the circumstance that previous mining operations, having lead for their object, had laid bare large amounts of zinc ore, ready for cheap extraction. It was a fortunate thing that the mines first opened at Joplin yielded rich blende, in large pure masses, associated with galena. For some years they were worked for lead. When a solid body of blende was reached the miners called it a "bar," and excavated—or, as the Western men would say, "gophered"—around it to save the expense of blasting it out. When the value of this ore was subsequently recognized, large quantities could be obtained from these bars, so rich as to need no concentration before shipment. Under less favorable conditions the development of the industry in this district would have been slow. Most of these bars appear now to have been exhausted. The zinc ore now brought into market comes mostly from deposits in the brittle flint rock, the fissures of which it fills. The whole mass extracted in mining must be crushed and

jigged. Galena is sometimes present and sometimes very scarce. Little is known of the extent of these deposits. They appear to be limited in height, width, and length, abutting against barren rock. At some points the zinc ore impregnates or interlaminates shales instead of flint. Iron sulphide is plentiful in a few mines only. Some of the ore consists of a skeleton of silica, as porous as a sponge, the pores of which are filled with blende; and all forms of transition are observed, from this to solid flint-rock, with disseminated specks of blende. This the miners call "iron-jack," their name for blende being "black-jack."

Next to the blende deposits of Joplin, the most important are the silicate mines at Granby. Here, also, the zinc ores were first exposed in large quantities by the mining of lead, and the earliest productiveness has considerably fallen off through the exhaustion of these reserves. The deposits of Granby seem to lie all in one horizon—either in one stratum of limestone, or in a group of several strata a few feet apart. The layers of ore-bearing rock are separated by barren layers.

Other deposits, again, resemble fissure-veins for considerable distances; but the local expression "run" designates most of them aptly. There is no foretelling how they will pitch or branch or turn.

At many of the mines the ore is broken by hand with the so-called buck-hammer on an anvil consisting of a hard rock, and is then dressed in Cornish jigs of the most ancient fashion, operated by hand with a pole. Stone-breakers and crushers are, however, coming into use, and several well-arranged concentrating works of large capacity have been built.

The attempt to build up a smelting industry on the spot, instead of shipping the ore out of the district, has not yet borne fruit. If, as now seems probable, the ore supply is to be for the most part the aggregate of many small and short-lived operations, scattered over a large area, the proper place for smelting-works is not at the locality of any one of these mines, but at the centres of transportation, where cheap freight and cheap fuel will command the whole field.

EXPERIMENTS WITH CHARCOAL, COKE AND ANTHRACITE IN THE PINE GROVE FURNACE, PA.

BY JOHN BIRKINBINE, PINE GROVE FURNACE.

In the spring of 1878 the Pine Grove Furnace, located in Cumberland County, Pa., was blown in after lying idle for several years. The furnace was constructed in 1770, and for over a century it has been in almost continual operation. The plant consisted in 1877, when the writer was first called there, of a stone stack 32 feet in height, inclosing a shaft and boshes, the latter being 8 feet in diameter. The blast was supplied by two wooden blowing-tubs discharging into a third, having a floating piston sustaining a box weighted to give the desired pressure, the power being furnished by a water-wheel. A small 18-pipe hot-blast stove heated the blast. The furnace was remodelled during the winter of '77–78, and a Weimer blowing-engine with a blowing-tub 5 feet in diameter and 2 feet stroke, and the necessary boilers, were substituted for the wooden tubs and water-wheel. The stack was raised and enlarged, and provided with a bell and hopper having a central drop. Water dam and tymp were added, and general repairs were made. The hot oven, however, was not increased. The furnace had always been operated with charcoal as fuel, and its reconstruction was made with a view to continue the use of charcoal, but provision was made for ample blowing capacity should other fuels at any time be employed.

Although liberal arrangements had been made in cutting wood, the stock of charcoal supplied in 1878 was insufficient for the increased requirements of the remodelled plant, and notwithstanding the purchase of some 80,000 bushels from an adjacent idle furnace, there was not enough to keep the furnace in constant blast until a new supply could be obtained, particularly as the season was unusually backward. Instead of following the established precedent of many charcoal furnace managers, *i. e.*, blow out every February, it was determined to continue in blast, using coke as fuel. Accordingly, when the charcoal stock was exhausted, on March 22d, 1879, Connellsville coke was substituted, the change of fuel being made at once; that is, the coke charges followed immediately upon the last charcoal charges. After working a few days, the strike in the Connellsville coke region cut off the supply, and anthracite coal was obtained, a mixture of the two fuels being employed. The strike continuing,

anthracite alone was used until a short time before a new supply of charcoal could be depended upon, when the coke shipments were resumed and mixed fuels again were charged. It is seldom that the results of the employment of different fuels in the same furnace under similar conditions are obtainable, and the following data are presented in hopes that they may appear opportunely for the discussion upon fuels, which has been participated in by Professor John A. Church, of New York, and England's metallurgical authority, Mr. I. Lowthian Bell.

The employment of other fuels than charcoal in a charcoal iron furnace is not presumed to be novel, and may have been experimented with at a number of furnaces. The only authentic information which has been procured is the following:

In the fall of 1853 anthracite coal was substituted for charcoal by the Messrs. Hunter, in the Moselem Furnace, Berks County, Pennsylvania, and its use continued for about four months. The furnace was 8 feet diameter at bosh and 31 feet in height, with open top, the blast being furnished by water power and heated in a stove. One tuyere was employed with charcoal, but a second tuyere was added when anthracite was used, these tuyeres being placed 22 inches above the bottom. From 35 to 45 tons of iron (mostly foundry) was made per week, with a consumption of from 2¼ to 2½ gross tons of anthracite. The output of the furnace was greater with anthracite than with charcoal, the yield with the latter fuel being from 26 to 35 tons per week, but the consumption of fuel per ton of iron was much greater with the anthracite. Mr. Nicholas Hunter, who kindly furnished the above information from memory, also states that 2¼-inch nozzles were used in each case, the tuyere being open when charcoal was used and closed when anthracite was charged. When making charcoal iron the lime charge was 15 to 20 per cent. of the ore charge; with anthracite this was increased to from 35 to 50 per cent. Unfortunately, the books of record of this furnace are believed to have been destroyed, and data in detail could not be obtained. It is, however, evident that the increased yield with anthracite was made by driving the furnace at the expense of fuel.

At the Philadelphia meeting in 1873, Mr. T. F. Witherbee presented a paper to the Institute upon the manufacture of Bessemer pig metal at the Fletcherville Charcoal Furnace, near Mineville, Essex County, New York, in which mention is made of the substitution of anthracite for charcoal towards the close of 1871. (See *Transactions*, Vol. II, pages 71 to 75.)

In 1873 Mr. S. M. Krauser changed the fuel in the Port Leyden Furnace, Lewis County, New York. The only record of this change which could be obtained is from a letter to the Secretary of the American Iron and Steel Association, viz., "We blew in with charcoal and made 409 tons; then filled with anthracite coal and made 697 tons; then changed to charcoal again, and we are still blowing."

Colonel George B. Weistling, a member of the Institute, had also used coke, but only for a short time, closing out a blast with charcoal at Mont Alto Furnace, Franklin County, Pa. It is to be regretted that full data of these experiments could not be obtained for comparison with those herewith presented.

The Pine Grove Furnace has a bosh 9 feet 4 inches in diameter, and a working height of 36 feet 6 inches; the tunnel-head is 5 feet in diameter, closed by bell and hopper, the former being 3 feet in diameter. The crucible is 50 inches in diameter and 5 feet in height, pierced for three tuyeres at a height of 3 feet from the bottom. There is no fore-hearth. The lining and bottom are of fire-brick. The temperature of the blast in all the experiments was between 500° F. and 700° F., the average being 600° F. It was nearly constant because, owing to the small size of the oven, it was continually worked to its utmost.

For some time previous to the use of coke as fuel, the charcoal used was mainly from the stock purchased from a neighboring idle furnace, and was deteriorated by the reloading, hauling by wagons and railroad, and the inclemency of the weather. The consumption, therefore, was above the proper working of the plant; nor was the output as great as it has been. To place results upon as equitable a basis as possible, comparison will be made with the operation of the furnace during the month of February, 1879, the last full month before the change in fuel was made, which was as follows:

Average blast per minute in cubic feet,	1,896.
Average pressure of blast in pounds per square inch,	0.77
Average weekly make of pig iron in tons (2260 pounds),	95.
Pounds of charcoal consumed per ton of iron,	2,531.
Cubic feet of air delivered per pound of charcoal,	77.8
Cubic feet of air delivered per ton of iron made,	197,084.
Tuyere area in square inches,	47.7
Average yield of ore per cent.,	38.26
Average per cent. of lime to ore charge,	22.
Ore and flux carried by one pound of fuel,	2.8

When the last charcoal was charged the furnace was working upon the following: 378 lbs. charcoal, 880 lbs. Pine Grove No. 1 ore,* 200 lbs. limestone. Number of revolutions of engine, 23, equivalent to 1806 cubic feet per minute. Pressure of blast, 0.6 lbs. through three 4½-inch nozzles. Average product per week, 90 tons. On March 23d, 1879, at 2 A.M., the charge was changed to 500 lbs. coke, 750 lbs. Pine Grove No. 1 ore, 375 lbs. limestone. As the coke descended in the shaft of the furnace there was no appreciable difference in the pressure of blast; and at 5 P.M., 15 hours after coke was first charged, the revolutions of the engine were increased from 23 to 28 without any increase of pressure. At 10 P.M. the revolutions were 32, and the pressure 1¼ lbs. The following day, owing to an accident to the water supply, two tuyeres were lost, and at that time three 3½-inch nozzles were substituted for the 4½-inch nozzles, which had continued in use up to this time. With the necessary trials to obtain best results, work could not be expected to be very regular; however, after the third day there was no serious trouble, and coke continued to be charged up to April 1st, when the record was as follows: Coke, 500 lbs.; Pine Grove No. 1 ore, 950 lbs.; limestone, 380 lbs.; revolutions of engine, 33, equal to 2592 cubic feet per minute; pressure, 1¼ lbs. through three 3½-inch nozzles. The strike of the Connellsville region necessitated that the coke be husbanded, and a charge of coke and anthracite coal was substituted. During the ten days coke alone was used the furnace received 687 charges, aggregating 346,000 lbs., or 8650 bushels of coke; 247.2 tons Pine Grove No. 1 ore; 109.7 tons limestone, and made 99 tons of iron, which, reduced to quantities per ton of iron, are equivalent to 3494 lbs. = 87.35 bushels coke; 2.50 tons Pine Grove No. 1 ore; 1.1 ton limestone; 323,845 cubic feet air = 92.66 cubic feet air per pound of coke consumed. Average pressure of blast, 1 lb.; maximum, 1¾ lbs.; 2.32 lbs. of ore and flux were carried by 1 lb. of fuel.

When it became necessary to use anthracite with the coke the tuyere nozzles were reduced to 2½ inches, and at 9 P.M., April 2d, the following charge was substituted: 500 lbs. anthracite coal, 80 lbs. coke, 950 lbs. Pine Grove No. 1 ore, 475 lbs. limestone. Revolutions of engine, 34 = 2670 cubic feet per minute. Pressure 1¾ lbs. through three 2½-inch nozzles. In this instance the action of the blast was quite different from that when the coke charges fol-

* No. 1 ore refers to ore from No. 1 bank; No. 2 ore to ore from No. 2 bank.

lowed those of charcoal, for, as the anthracite descended in the shaft of the furnace, the pressure increased.

The following is a record of the pressure gauge April 2d and 3d, 1879:

April 2d,	9 P.M.	Revolutions, 34	Pressure, $1\frac{3}{4}$ lbs.
"	10 P.M.	" 34	" 2 "
April 3d,	1 A.M.	" 34	" $2\frac{1}{4}$ "
"	3 A.M.	" 34	" $2\frac{1}{2}$ "
"	8 A.M.	" 34	" $2\frac{3}{4}$ "
"	10 A.M.	" 34	" 3 "
"	12 noon.	" 34	" $3\frac{1}{4}$ "

It continued at this pressure until 5 P.M., when the revolutions were increased. A careful estimate showed that the anthracite would be at work at 2 P.M., but the maximum pressure at 34 revolutions was reached at noon. The increase due to the resistance of the stock was, therefore, $1\frac{1}{2}$ lbs. greater with anthracite mixture than with coke alone. Up to this time no changes had been made in ores, nor were any made until April 6th, when a mixture of ores from the Pine Grove banks, Nos. 1 and 2, and magnetic ore from the Fuller mine, near Dillsburg, were employed. For ten days up to April 11th the same mixture of fuels was continued, but the impossibility of obtaining coke necessitated a dependence upon anthracite alone. During these ten days the engine had been run at an average of 35.5 revolutions and an average pressure of $4\frac{3}{4}$ lbs.

The following is a record of the work with mixed fuels, the average mixture being 81.5 per cent. anthracite, 18.5 per cent. coke: 322,600 lbs. anthracite and 73,160 lbs. coke, making a total of 395,760 lbs. fuel consumed in making $112\frac{1}{2}$ tons of iron; 292.3 tons of mixed ores and 145.26 tons limestone were charged in this time. Therefore, to make a ton of iron the following quantities were required: 3473 lbs. of fuel, 2.6 tons of ore, 1.29 tons limestone, 338,187 cubic feet of air = 96.13 cubic feet per lb. of fuel; 2.47 lbs. ores and flux carried with 1 lb. of fuel. When the last charge of mixed fuels was put into the furnace the charge was changed to 600 lbs. anthracite, 860 lbs. Pine Grove No. 2 ore, 90 lbs. magnetic "Fuller" ore, 525 lbs. limestone. Revolutions of engine, 34; pressure, 4 lbs. through three $2\frac{1}{2}$-inch nozzles. During the time the anthracite was descending through the furnace the pressure increased to $4\frac{1}{2}$ lbs., demonstrating how a small amount of coke (one-seventh) had aided in keeping the furnace open. With no extraordinary disturbances the furnace continued on anthracite alone as fuel from April 12th to May

4th, a period of twenty-three days; there were some changes in burden and volume of air, but the quantities were in general constant, except that the limestone was reduced to 475 lbs. and No. 2 ore only was used; the records of the two days mentioned being identical. During this time, however, about 10 tons of scrap, which had accumulated about the plant, was charged into the furnace, and in all estimates it is taken as 90 per cent. ore.

The record of the twenty-three days' blast with anthracite as fuel is: Average pressure of blast, 4¼ lbs.; maximum, 5½ lbs.; 1236 charges aggregated 331 tons coal, 480 tons ore, 227.6 tons limestone, 10 tons scrap, which produced 191½ tons of pig iron, and gives the following quantities per ton of pig iron: 3871 lbs. (1.728 tons) coal, 2.63 tons ore, 1.19 tons limestone, 398,679 cubic feet of air = 103 cubic feet of air per pound of coal, 2.14 lbs. of ore, scrap, and flux, carried by 1 lb. of coal. Average per cent. of ore, 38; average lime burden, 47.4 per cent.

From May 5th to May 23d coke was added to the anthracite, shipments having been resumed, and at the latter date, the anthracite stock being exhausted, coke alone was charged for two days, until that, too, was used up, when the furnace returned to charcoal as fuel, which it has continued to use to the present time.

On May 5th the charge of the furnace was made 125 lbs. coke and 450 lbs. anthracite (instead of 600 lbs. anthracite, which had been the basis up to this time), 950 lbs. Pine Grove No. 2 ore, 475 lbs. limestone. Revolutions, 34; pressure, 4¾ lbs. Although no change was made in the speed of the engine, the pressure gradually fell to 3¾ lbs.

On May 10th the proportions of the fuel were changed to 285 lbs. anthracite and 285 lbs. coke; the pressure gradually fell until at 34 revolutions it was 2½ lbs. The working of the furnace necessitated a reduction of burden to 850 lbs. Pine Grove No. 2 ore and 425 lbs. limestone, the revolutions being reduced to 30 and the pressure to 1½ lbs. Several minor changes in the proportions of coke and anthracite were made to clean up stock, and on May 22d the charge was made 500 lbs. of coke, until it, too, was consumed. The operation of the furnace working for the last two days on coke alone showed a marked decrease in pressure, which at 32 revolutions was 1¾ lbs. Taking the entire period from May 5th to May 23d as working on mixed fuels, the following results are obtained: 1171 charges were hoisted, aggregating 161.4 tons anthracite, 135 tons, or 7560 bushels of coke; 460.36 Pine Grove No. 2 ore, 3 tons scrap iron, 230.16 tons

limestone; and 203 tons of pig iron were made, which reduced to quantities per ton of iron give: 2.3 tons of Pine Grove No. 2 ore; 1.13 tons of limestone; 1.46 tons of fuel = 3217 lbs. (55 per cent. of anthracite and 45 per cent. of coke); 317,170 cubic feet of air = 99.3 cubic feet per lb. of fuel; 2.35 lbs. ore and flux carried by 1 lb. of fuel. Average pressure of blast, 2½ lbs.; maximum, 4¾ lbs.

The anthracite coal used was medium soft and of "steamboat" size. In dividing the various casts of pig iron and crediting them to the different fuels, due allowance was made for the "driving" of the furnace. In estimating this a given weight of charcoal was assumed as occupying double the space of the same weight of coke, and four times the space of the same weight of anthracite. Allowing for difference in ore and lime burdens, a practically correct estimate was thus made. The most rapid driving of the furnace during February, when running on charcoal, was 12½ hours; when running on all coke, the best day's record showed the stock to have been 20½ hours in the furnace; when using 81½ per cent. anthracite and 18½ per cent. coke, the fastest driving was 22 hours; while with anthracite alone it was 31 hours, and with 55 per cent. anthracite and 45 per cent. coke it was 25 hours.

During the month of August, 1879, the furnace "drove," so that stock did not remain in it but 9½ hours—this was with charcoal as fuel. There was no attempt to make a gray iron for foundry purposes, as the market for the charcoal pig iron is entirely confined to charcoal forges, and for the iron made with anthracite, coke or mixed fuels the demand was for mill iron. The records here given are not offered as extraordinary, nor is it claimed that experiments over such brief periods can establish any fair ratio of value for the different fuels. The first ten days during which coke was used were partially consumed in trials, and are therefore unfair to this fuel, and in none of the experiments was the furnace run for a sufficient time to make an equitable comparison. No allowance for leakages was made in calculating the air consumption, and the results obtained are presented more for comparison than to demonstrate the actual amount of air used. There were no defective parts, and the leakages were only such as exist in any furnace in good working order, and those occasioned by snuffing tuyeres, opening notches, etc., which are difficult to determine.

The following table is merely a recapitulation of data heretofore given, placed in a convenient form for investigation. All the figures given are averaged for the time the fuel mentioned at the head of

the columns was used. The tons are 2260 lbs. each for pig iron, 2240 lbs. for everything else:

SUMMARY OF RESULTS OF SMELTING WITH CHARCOAL, COKE, MIXED FUEL, AND ANTHRACITE AT THE PINE GROVE FURNACE, CUMBERLAND COUNTY, PENNSYLVANIA.

ITEMS FROM RECORD OF WORKING.	Charcoal, February, 1879.	All coke, March 22, to April 2, 1879.	81.5 per cent. anth., 18.5 per cent. coke, April, 1879.	All anthracite, April and May, 1879.	Anth. 55 per cent., coke 45 per cent., May, 1879.	Charcoal, August, 1879.
Pounds of fuel consumed per ton of pig iron.	2,531	3,494	3,473	3,871	3,271	2,650
Pounds ore and flux carried per lb. of fuel.	2.8	2.32	2.47	2.14	2 34	2.64
Tons of iron made per week, average	95	70	78	58	77	101.6
Percentage of iron yielded in furnace	38.26	40	38 40	38	43.3	40
Percentage of lime to ore burden	22	44	50	47.4	50	24.4
Average cubic feet of blast per minute.	1,896	2,435	2,749	2,434	2,473	2,301
Average cubic feet of air per ton of iron	197,084	323,845	338,187	398.679	317,170	216,243
Average cubic feet of air per lb. fuel	77.8	92.66	96.13	103	97	81
Average pressure of blast in pounds	0.77	1	4 25	4 75	2 5	1.25
Tuyere area in square inches	47.7	28.9	14.7	14 7	14.7	42.5
Least time that stock was in furnace, hours.	12.5	20.5	22	31	25	9.5
Grade of iron	2.4	3	2 7	3	3	2
Duration of experiments, days	28	10	10	23	19	31

DISCUSSION.

DR. CHURCH: Mr. Birkinbine has taken advantage of a rare combination of circumstances to give a much-needed precision to the relation of the three fuels, charcoal, coke, and anthracite, when burned under the peculiar conditions of the blast-furnace hearth. His observation, upon the increase of pressure in changing from coke to hard coal, and the absence of increase when one of the lighter fuels was changed for the other, will be especially valuable if it suffices to relieve this simple part of the blast-furnace practice and theory from the absurd explanations which have been thrust upon it. Another of his minor observations is one to which I once gave a good deal of attention. That is the quantity of blast entering the furnace with different pressures. I long ago reached the conclusion that the leakage with high pressures was so great as to vitiate those conclusions concerning the blast which are drawn from the work of the blast engine exclusively, as in former times it was the wont to draw them. But I have been most interested in the comparative consumption of the different fuels, and I think that the results of Mr. Birkinbine's enforced experiments are confirmatory of that theory of combustion in the hearth which I have advocated before the Institute. With his figures before us, showing how much greater the consumption of

anthracite than of charcoal was, it is difficult to imagine that all this extra carbon could have been burnt in the hearth. Had that taken place it seems rational to suppose that the furnace would have driven much faster with hard coal than with soft.

Of course the consumption of air increased with that of coal, and the interstitial spaces in the mass of anthracite and mine are necessarily less than with an equal bulk of charcoal and mine. Thus the velocity of the blast through the hearth was increased both by the increase of its quantity and the reduced size of its paths. Its stay in the hearth was much shorter, and if we admit that the factors of combustion in confined spaces are surface of contact, strength of the oxidizing agent, and time of contact, it is apparent that one, and I hold that two, of these factors were lessened by changing from charcoal to anthracite. The one that was certainly altered was time. The larger quantity of blast had to be driven more quickly through the zone of fusion, and even if the quality of the fuel had not been changed, the consumption per ton of pig metal would have increased if more blast had been put on than that quantity which gives a *maximum* product with a hearth having the dimensions of that in question. That increase I hold to be due to the fact that the time of passage through the hearth is too short to allow the abstraction of all the oxygen, even with a fuel so combustible as charcoal. Some remains to burn above that zone, and in a part of the furnace where, though it produces heat, the only effect is to excite to abnormal energy one of the heat-consuming operations which go on in the body of the furnace.

The other factor, which in my view is decreased in force by a change from soft to hard fuel, is the strength of the oxidizing agent. Of course if less oxygen is taken up in the hearth, the gas is more oxidizing than when the normal quantity is burned. But all of these values are relative. A mixture of nitrogen and oxygen, which is strongly oxidizing towards the porous and inflammable charcoal, may have much feebler action upon the dense and close-built anthracite, so that the increase of blast must be sufficient not only to maintain a state of oxidation energetic enough to do equal work in less time, but also equal work with a less docile reagent. The gas reaches the limit of the zone of fusion with unconsumed oxygen remaining in it, and this surplus so increases the heat of the upper region that the operations there are greatly stimulated. One of them is the reduction of carbonic acid, which tends to counteract the heating effects of the surplus oxygen.

Considering the relative action of charcoal and anthracite in the hearth and their effect upon the blast, we should conclude under this explanation, that the tunnel gas of a hard-coal furnace should be hotter and contain less carbonic acid than that of a charcoal furnace, and that among anthracite furnaces those which use most fuel and blast should discharge the hottest and least oxidized gas.

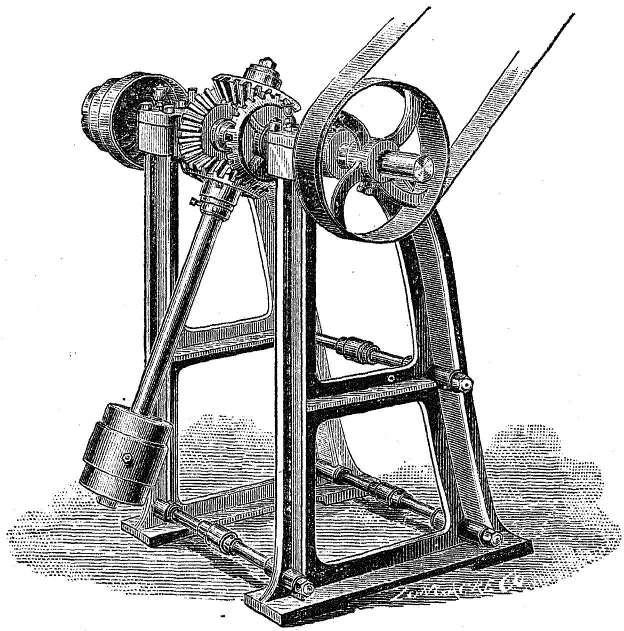
AN AUTOGRAPHIC TRANSMITTING DYNAMOMETER.

BY WILLIAM KENT, M.E., PITTSBURGH, PA.

THE dynamometer herein described is a modification of the one invented by Mr. Samuel Batchelder, of Boston, nearly forty years ago, a description of which may be found in the *Journal of the Franklin Institute*, 1843, vol. xxxii, p. 277, and in the *Scientific American* of August 31st, 1878. The modification consists in providing a method of making an automatic record, and of indicating more minute variations of the power transmitted. The improved dynamometer was designed by the writer in 1877, and was built a few months since by the Class of 1879 of the Stevens Institute of Technology. The accompanying cut, taken from the *Annual Announcement* of the Stevens Institute, represents the dynamometer without the recording attachment. It consists, as shown, of two stout cast-iron frames, held together by bolts in bearings, in the top of which frames run two shafts, each carrying a pulley at its outer end and a bevel gear-wheel of 45° at its inner end. One of these shafts is the driving-shaft, connected by belt to the engine or other prime mover; the other is the driven shaft, connected by belt to the machine driven. The power is transmitted from one shaft to the other through two other bevel-wheels of 45° gearing with the first, the shaft common to them and on which they run freely being at right angles to the axis of the two shafts first mentioned, and carrying at one extremity a heavy pendulum.

The bevel-wheels being connected, as shown, and the power being applied to the driving-wheel, the two intermediate wheels with their common shaft have a tendency to revolve around the driving-axis, which tendency is a measure of the force transmitted, and is resisted by the moment of the weight of the pendulum. In the Batchelder dynamometer the four bevel-wheels and their shafts are used, but

the shaft connecting the intermediate wheels is always held in a horizontal position, and its tendency to revolve is resisted by weights and a sliding poise applied to an extension of one end of it, which is graduated like a scale-beam. In using the Batchelder dynamometer the operator requires to keep the beam constantly balanced, by shifting the poise on the scale-beam, or the weights in the scale-pan hung at its outer end, to correspond with the variations of the power transmitted, and a record of the power is obtained by noting the weight on the scale-beam at each instant, and the corresponding number of revolutions of the driving-shaft. The horse-power is



obtained by multiplying the force in pounds registered on the scale-beam by the distance in feet of its centre of gravity from the driving-axis, by the number of revolutions per minute, by 3.1416, and dividing by 33000. In the improved dynamometer the horizontal scale-beam with its weights and sliding poise have been dispensed with, and the swinging pendulum substituted. The tendency of the shaft carrying the two intermediate bevel-wheels, the prolongation of which shaft is the pendulum arm, to revolve around the driving-axis, is measured by the weight of the pendulum and its arm multiplied by the distance of their centre of gravity from the driving-axis and by

the sine of the angle which the pendulum arm makes with the vertical. When the pendulum hangs in a vertical position the force transmitted is zero, errors due to friction excepted, and when it is horizontal, the sine of the angle being equal to unity, the force is the maximum the apparatus is capable of recording. The weight of pendulum and its position on the arm being constant, the only variables to be considered in measuring the horse-power transmitted are the number of revolutions per minute and the sine of the angle of the inclination of the arm. These variables may be caused to automatically record themselves on a sheet of cross-section paper by any one of a number of devices. The plan designed by the writer is shown in the diagram accompanying this paper. A quadrant of metal, A, with a groove in its outer edge, is adjusted to the pendulum arm in such a position that its centre is in the line of the driving-axis. A cord, B, is fastened to the lower extremity of this grooved quadrant in such a way that it will be wound upon the groove as the pendulum moves in the direction of the arrow. This cord is carried off at a tangent over one or more pulleys and to a pencil, C, moving in a slide across the breadth of a long piece of suitably ruled diagram paper, D, which may be wrapped on a drum, or rolled from one drum on to another. As shown on the Plate the longitudinal lines upon the diagram paper are ruled in such a manner that their distances apart are proportional to the sines of the angles of inclination of the pendulum from 0° to 90°. The vertical lines are ruled equidistant, their distance apart representing any given number of revolutions. The drum upon which the paper is placed is geared to the driving-shaft, so that its rate of travel will be very slow, but proportional to the speed of the shaft. A clockwork attachment may be arranged so as to make a mark upon the paper at the end of each minute or other period of time, so that the diagram made by the pencil when the apparatus is in motion will be a record of force, number of revolutions, and time.

By an examination of the scale at the end of the diagram paper, it will be seen that the principle of recording the sines of the angles of inclination of the arm gives a method of recording exceedingly small percentages of the power transmitted when the total variations in that power are not excessive. Let us suppose that a machine was being tested, in which the power necessary to drive it varied only between 9½ and 10 horse-power, or a little over 5 per cent. The pendulum might be shifted on its arm to such a position that the pencil would trace a line on the diagram paper between the lines marked .9500 and

1.0000, and the inclination of the arm would vary correspondingly from 71° 48′ to 90°, or nearly 20 per cent. of its total range. If the diagram paper was 10 inches wide, and measurements could be made to $\frac{1}{50}$th of an inch, an average variation between the ranges of $9\frac{1}{2}$ and 10 horse-power of .005 horse-power, or .05 of one per cent. could be detected and measured. If the total variation of the power transmitted (say 10 horse-power) is as low as 1 per cent., then by causing the diagram to be traced between the lines .9900 and 1.0000, a variation of .01 of one per cent., or .001 horse-power could be detected.

The dynamometer at the Stevens Institute has a capacity for measuring 20 horse-power. A method is provided of measuring very small powers, which consists in lessening the moment of the pendulum and arm, first by shifting the sliding weight nearer the driving-axis; second, if still lighter moment is desired, by removing the weight from the arm entirely; or, third, if even still greater delicacy is desired, by counterbalancing the weight of the arm by adding weight to its upper end, above the upper intermediate bevel-wheel.

The writer has also designed an application of the swinging pendulum and automatic recording attachment to the friction dynamometer, well known as the Prony brake. In this dynamometer the power is not transmitted, but converted into heat by means of the friction of a stationary tightened band around a revolving wheel. The power is measured by weights applied in a pan at the extremity of a horizontal beam, as in the Batchelder dynamometer. In the modification, the horizontal beam and weight pan are dispensed with, and the swinging pendulum with its autographic registry substituted, just as in the improvement on the Batchelder dynamometer above described.

RELATIONS OF SULPHUR IN COAL AND COKE.

BY DR. JAMES P. KIMBALL, LEHIGH UNIVERSITY, BETHLEHEM, PA.

SULPHUR is always present in mineral coal of every variety. In the oxidized state it may exist as sulphuric acid in combination with a base. In the unoxidized state it exists in combination with iron as bisulphide of iron. It is also supposed by some chemists to exist in unknown combinations with the organic elements of the coal. This supposition requires the demonstration as a fact that certain coals yield more sulphur than what may exist in a free state, or what belongs to recognized combinations, not only with the accompanying iron but with other inorganic bases abundantly occurring in coals, but which unfortunately are very rarely estimated. This practically as well as theoretically important question it is at present impossible to decide, simply for want of available thorough ultimate coal analyses including the ash. Such analyses, at the best imperfect, are laborious and intricate, and therefore seldom made, the much simpler proximate analyses, so-called, being resorted to by most chemists for technical purposes, and commonly substituted for more scientific work even in the laboratories of government surveys. And it is unfortunate enough that few ultimate analyses are extended to the composition of the ash, much less to its relation to the original constitution of the coal.

Sulphur chiefly occurs in most coals in the form of yellow pyrites (pyrite) and white pyrites (marcasite), both minerals, characterized by differences in crystalline form and in physical properties as well as in color, but having the same chemical composition (FeS_2). No coal is entirely free from one or both of these dimorphous forms of bisulphide of iron. It may be disseminated through the mass so as to be invisible, in plates or scales between the laminæ of the coal, or in conspicuous segregations or nodules, sometimes of considerable dimensions.

Coals greatly differ as to the proportion of sulphur which they contain. They also differ as to the relative proportion in which its several forms of combination occur. Such qualitative and quantitative disparities are scarcely, if at all, more marked in separate coal seams of a uniform type than in different parts of one and the same coal seam, whether it be considered locally as constituted of several

layers or "benches," or whether considered in its horizontal identity over an extensive territory. Nor is any restriction to uniformity of type necessary; for no single type of mineral coal has been found to enjoy an immunity from sulphur, or in respect to this unwelcome constituent in some of its several forms any perceptible advantage over kindred types. An excess of sulphur in the prevailing form of pyrites may be mainly confined to certain parts of a coal seam—to an upper or lower bench, as the case may be, or to an intermediate position. Again, it may take possession of more than one of such divisions of a coal seam. Wherever thus found concentrated and well defined, it becomes practicable to reject it in mining. Indeed, it is customary to regard such occurrences in coal mines as of no little practical benefit.

On the other hand, a coal seam, either wholly or in part, may be without any such concentration of visible sulphur-bearing compounds, but contain them more or less irregularly diffused throughout the whole mass.

It will, therefore, readily be perceived that the estimation of the quantity of sulphur in coal by analysis depends more upon the selection of the sample than the nicety of the chemist. The most important consideration which attaches to a skilful analysis of any kind is, after all, what is the real significance of the sample? And yet this consideration is one which does not always suggest itself. The question whether a sample be an *average* of whatever it is designed to test by chemical analysis, is often raised too late to be answered. It would be a safe rule to regard all analyses as indecisive, the mode of sampling which is not satisfactorily accounted for. This is only a due regard to the personal equation, and to the propensity in man to take the best.

The deviations between different parts of a single coal-bed, in respect to the state and relative quantity of sulphur, are sometimes of no wider range than in respect to the other organic as well as inorganic constituents. In mining practice a choice is often made between different parts of a coal seam, by the total or partial rejection of certain parts either inside the mine or in preparing the coal for use. An *average* of the whole of a seam may therefore be anything but a practical sample. Among the many mistakes of omission and commission in the practical estimation of coal, not a few will be found on this side of the difficulty of coal sampling.

In several branches of commerce depending on assay values, due

importance is given in sampling to the personal equation. For while both buyer and seller in certain transactions may be agreed upon the services of the same chemist, sampling in the interest of each party is commonly held to be indispensable. Even if such observances be nothing more than ordinary business precautions, the practice commends itself as at least a practical recognition of the liability to involuntary error on the part of the sampler. Indeed, sampling with the object of obtaining an average specimen of substances of unequal composition, like ores of the precious and some of the base metals, to the commercial averaging of which the above allusion is made, is universally recognized as an operation demanding both care and skill. But it is the exception rather than the rule to apply similar painstaking methods to the sampling of iron ores and coals. Analyses of these minerals too often mislead for want of the same degree of care in sampling that is bestowed on less bulky minerals. Everybody knows of serious industrial mistakes arising from so-called "fancy analyses." It is safe to say that in ninety-nine of a hundred such cases the epithet really belongs to the sample.

While descriptions of methods of analysis are not unfrequently met with in scientific publications, seldom is any satisfaction afforded to the reader as to the method or the thoroughness of the sampling, though of equal importance with the finer work of the laboratory. The urgency of the above remarks will appear when it is reflected how minute is the quantity of almost all mineral substances actually used in the delicate manipulations of the analytical chemist. With reference to what has been said on this subject, the present geological survey of Kentucky furnishes a commendable example as well as wise and timely precept.*

The concentration of pyrites in coal seams is often seen to be more or less affected by local topographical or rather stratigraphical circumstances. Thus its distribution in a conspicuous form is sometimes found to be limited to the depressed portions of basins, or to certain slopes of anticlinal or synclinal undulations of the coal measures. Portions of a usually sulphurous coal seam may be found to prove low in sulphur; while, conversely, portions of a seam usually low in sulphur as often disappoint the miner by high ratios of that substance. In such cases we still observe the prevalence of

* Geological Survey of Kentucky, New Series, vol. i, pp. 148, 328, 340, 401.

the fact of concentration of sulphur in at least the form of pyrites, in obedience to the laws of circulating subterranean waters and molecular forces which govern the distribution of sulphur in coal in all the ways here instanced, and in many more besides.

Although but few analyses reward the quest of such as have been carried far enough to bear upon the point, an examination of recent tables of the more searching chemical investigations of coals will not fail to show numerous exceptions to the prevailing occurrence of sulphur in coals in the binary form of bisulphide of iron. It seems indeed by no means certain, that in the most common, or bituminous, type of coal, such exceptions are not sufficiently in force to unsettle the general belief in the predominance of this conspicuous compound of iron and sulphur in all coals above any other mode of occurrence of sulphur.

A great majority of even the best coal analyses on record fall short of discriminating determinations of sulphur however complete the estimation of its aggregate. It may be questioned whether the present conventional methods of coal analysis are not without recognition of the fact that even freshly mined coals sometimes contain soluble sulphates, which in an early stage of the analysis should be extracted and specially determined. As sulphates of iron are among these, it cannot always be correct to assign all the iron of the ash on the theory of its immediate existence in the coal as bisulphide. Nor is it safe to conclude, as lately in more than a single instance, that an excess of sulphur over what the iron of the ash demands to form bisulphide, must be referred to combination with organic parts of the coal, especially with lime, magnesia, or alumina, or perhaps with all of these bases in the ash, calling at least for some assignment of sulphur.*

The common practice of estimating sulphur from its determination as sulphuric acid, from which the amount corresponding to the sulphates of the ash is deducted, is open to the objection that to a variable extent this amount may be increased by the conversion of some portion of the sulphur to sulphuric acid in the process of incineration, thus lessening perhaps in the presence of lime and magnesia the proportion of sulphur to be accounted for. Except when the stable sulphates of these bases were originally present in the coal, correction

* See Geological Survey of Pennsylvania, Report M, p. 31; Geological Survey of Ohio, Annual Report II, 1870, p. 108.

for an amount of sulphur equivalent to the sulphates may sometimes be important. When the bisulphide of iron in coal is in the unstable form of white pyrites, its rapid oxidation, under favorable circumstances, establishes differences in the state of the sulphur from day to day, if not indeed, under favorable circumstances, within very few hours.

A series of proximate analyses by Professor Wormley of coal from the well-known Straitsville or Nelsonville seam of Ohio, on Lost Run, show in no instance the presence of sufficient iron to satisfy the percentage of sulphur for the formation of bisulphide of iron. In the absence of analyses of at least the ash, it is impossible to account for the excess of sulphur, but for the reason already indicated, if for no other, it were rash to conclude with Professor Andrews and Mr. McCreath that a part of it is in permanent combination with the fixed carbon of the coal.*

Another sample of Ohio coal which for fourteen years had lain in a college cabinet, and which exhibited no bisulphide of iron nor any products of the oxidation of this substance, yielded to the same chemist 2.885 per cent. of sulphur in excess of what the percentage of iron (0.39) required to form bisulphide, viz., 0.445.† Mr. McCreath, of the Second Geological Survey of Pennsylvania,‡ ob-

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* Geological Survey of Ohio, Annual Report, II, 1870, pp. 105–108; also Annual Report for 1870, p. 413, Second Geological Survey of Pennsylvania, Report M, p. 30.

† Geological Survey of Ohio, ibid., p. 108.

‡ Since the preparation of this paper the appearance of Report MM of the Second Geological Survey of Pennsylvania has supplied a number of instructive analyses of coals and cokes, at the hand of Mr. McCreath, chemist of that survey. All of the coals examined from different parts of the State, according to Mr. McCreath, contain a proportion of sulphur in excess of what is required to form bisulphide of iron, varying from 1.95 to 75.25 per cent., or an average of 33.79 per cent. of the total amount of sulphur present. On general grounds, for the reasons above given, such excess of sulphur can hardly be pronounced "free sulphur" without reference to other bases in the ash besides iron. The fact that this excess of sulphur is not expelled in the process of coking to any considerable extent, as shown by Mr. McCreath, points, it seems to me, to its combination in the several coals with lime, magnesia or alumina, or perhaps in some cases with more than one or with all of these bases. Strictly free sulphur in the presence of carbonate of lime would doubtless become fixed in combination with the lime in the process of incineration. Rather than a reference of the excess of sulphur to its existence in a free state, the more probable conjecture would seem to me to be that a large portion of it is in the form of sulphuric acid in gypsum, originally present, or else the result of a combination brought about by the con-

serves a similar disproportion between the sulphur and iron found in a number of coals of that State, and cites one example of coal from Jefferson County which was found to contain an excess of sulphur amounting to 3.53 per cent.* Dr. Peter, of the Second Geological Survey of Kentucky, has carried a few analyses far enough to prove that sulphur is sometimes in combination in the coals of that State as calcic sulphate, and that it also exists as strictly free sulphur. In a specimen of fibrous coal, this chemist detected 3.632 per cent. of sulphate of lime together with a certain amount of uncombined sulphur.† The determination by Mr. A. A. Blair of the relative amounts of iron and sulphur in a number of Missouri coals, was attended with similar results to those previously obtained by Professor Wormley, and those since obtained by Mr. McCreath. One coal from Lincoln County is stated to have yielded 2.632 per cent. of sulphur without traces of iron.‡

Upon partially investigating the relations of sulphur in a collection of Iowa coals the chemist of the Geological Survey of that State was led to the following results:

Water slightly acidulated with hydrochloric acid extracted considerable proportions of soluble matter in addition to sulphates of iron. Four of the ten specimens examined proved to be rich in lime, especially in its carbonate, and another poor in this base, which in each case appeared to be present also to some extent as sulphate. In a large number of coals from the same State, tested by Mr. Emery for calcic carbonate, but few failed to show its presence. In four of the ten coals first named, the relative proportion of iron and sulphur was determined to be nearly that existing in bisulphide of iron. In four others a marked excess of sulphur was discovered. In two of the number a deficiency was noted, probably attributable to the formation of ferrous sulphate and elimination of excess of sulphuric acid. This acid was found in the whole series of coals.§

Analyses of the ashes of coals, scarcely without exception, show

ditions of the analysis. It is not questioned, however, that an appreciable proportion of such an excess of sulphur may exist in a strictly free state, as found to be the case with certain Kentucky coals. The large proportion of sulphur retained by the coke from some of these coals is, however, against the assumption of this being the case in more than a few of the examples recorded in Mr. McCreath's valuable table, p. 124. (See subsequent footnote, page 193.)

* Report M, p. 31.

† Geological Report I, New Series, p. 287.

‡ Geological Survey of Missouri, 1872, p. 36.

§ Am. Jour. of Sci., iii, 34.

the presence of lime in proportions varying from the fractional part of a unit to upwards of 14 per cent., according to the tables prepared by Percy,* while lignites generally show a much higher proportion of this base, varying in round numbers from 15 to 45 per cent.

As lime is generally estimated in the ash after incineration of the coal, which must often result in an increase of the calcic sulphate, whatever its previous different state, conventional analyses of coal ashes leave a doubt as to the original state of combination of the lime. It is natural to suppose, however, that while a portion of it is present as a component of silicates belonging to the miscellaneous sedimentary matter of coal, and another portion as carbonate, a greater or less proportion—according to the excess of sulphur above the requirements of the iron—must be originally present as calcic sulphate. It would at least be unsafe to conclude that any such excess of sulphur may be in combination with organic constituents of the coal, without previously ascertaining by direct method the proportion of lime and other bases present as well as their actual constitution.†

If the sulphates of lime and magnesia played so important a part in the marine sedimentation of coal in conjunction with decaying organic matter as to have chiefly furnished the sulphur of the pyrites found in coal, their presence in coals as products of epigenesis,‡ and in inverse proportion to their widely different degrees of solubility, seems natural enough to suppose. In the absence of analyses conclusively bearing on the question of the general diffusion of these sulphates in mineral coals, it may be conjectured that, as a rule, in sulphurous coals not sensibly pyritic the greater part of the sulphur will be found with lime as a sulphate. From available analyses but imperfectly bearing on this point, and from the apparently universal presence of lime in coals, often in large proportions, it seems probable that in combination with sulphuric acid, and as, therefore, a compound of sulphur, it is only second in importance to pyrites. Comparative irregularities in different modes of distillation and combustion of coals, as to the relative elimination and retention of sulphur, are doubtless in many cases to be referred to the action of lime in gypsiferous or calcareous varieties, either by fixing sulphuric acid or retaining its own complement of it. In such cases, whether it be in the form of carbonate or sulphate, it must, in full measure of the presence of sulphur, have a good or bad effect on the practical qualities of the coal ac-

* Percy's Fuel, 351-353.

† See Percy's Fuel, p. 326.

‡ Hunt's Chem. Essays, pp. 87, 99, 230.

cording to the function performed by it in its different uses in the arts.* If this be so, the quantitative estimation of at least this principal base in the ash may be in certain cases of practical importance, as affording some light at least on the relative proportion of separate compounds of sulphur.

Magnesia is also found amongst the sedimentary matter entering mechanically into the composition of mineral coal. To whatever extent this may be in special cases, it is not improbable that a further small proportion as a part of the inorganic remains of aquatic vegetation generally enters into the actual constitution of coal.† Its state of combination in unweathered coal often appears to be that of silicate. In the presence of calcic carbonate, it might be supposed to have to some extent a dolomitic relation. As a common ingredient of sea-water and mineral waters, magnesic sulphate may sometimes be an indirect but hardly a direct deposit in coal. It might also be supposed to occur in the same way by the action of water saturated with gypsum on dolomitic matter in coal or on magneso-calcareous strata intercalated with it. This reaction is, however, called in question by Hunt.‡ In weathered coal it often exists as a sulphate through double decomposition of its silicates with bisulphide of iron. From the extreme solubility of the sulphate (epsomite), which appears as an efflorescence upon exposed surfaces, it often escapes through the action of circulating waters as soon as formed.

It is hardly to be supposed that lime, whether as carbonate or sulphate, is generally present in carboniferous coals to any such extent as in the few examples of Kentucky coals reported by Dr. Peter. Yet so often does even an ultimate analysis of coal stop short of an analysis of the ash, that as a frequent occurrence in coals lime may have been generally overlooked. Facts of the kind already adverted to indicate the probability that coals containing lime salts to an important degree are more widely distributed than at present actually recognized. Again, certain parts of the coal measures prove more calcareous than others, especially, for instance, in the interior basins of the great Allegheny coal-field, where the coal measures themselves contain more frequent intercalations of limestones, some of which are more or less ferriferous. That calcareous strata rela-

* Probably a bad effect of lime in coal is experienced only in ordinary firing, when it disposes the formation of clinker. Its presence in coal for smelting purposes and for gas-making, is rather an advantage than not, as will presently be shown in the text.

† Bischof, Chem. Geol., i, 611.

‡ Chem. Essays, p. 106

tively occupy a much larger part of the palæozoic series in the interior of the continent than in the eastern border region is a well-known geological fact. Stratigraphical as well as geographical differences of this kind likewise suggest themselves. Calcareous waters arrested within a coal seam by means of an impervious under-clay, or even circulating freely within one, would deposit calcic sulphate. Such waters are sufficiently charged with atmospheric air to perform the function of oxidation at greater depths than are freely penetrated by the atmosphere itself. Deepseated chemical reactions depending on oxidation and solution are doubtless often greatly facilitated by the intermittent or alternate access of air and water.

Sulphur in coal may be separated by the oxidation of the hydrogen of hydric sulphide in solution in the waters of coal-beds. This gas is probably generated in coal seams by the reduction of calcic sulphate to calcic sulphide in the presence of decaying organic matter, and by the action of carbonic acid upon the resulting calcic sulphide. The reduction of magnesic sulphate by organic matter, and the decomposition of the sulphide by carbonic acid, must in a similar manner result in the generation of hydric sulphide.

The sulphur observed in the fibrous coal or mineral charcoal of several seams in the lower coal measures of Kentucky is disposed in flakes between the laminæ, without concentration. Another indication of the above origin is its state of fine subdivision.

The number of compounds of sulphur in coals is multiplied by epigenesis of its prevailing forms under various influences, such as are exerted by other accidental constituents, and by exposure to the atmosphere, to water, and to extraneous matter dissolved in it; and also, it may be added, by elevation of temperature. As products of a succession of chemical changes from a more or less uniform beginning, coals essentially differ among themselves according to the different degrees in which these changes have been accomplished. But it is not always practicable to distinguish between changes belonging to this natural succession, so far as they appear regular, and such as result from artificial conditions, natural vicissitudes, or from accidental or non-essential properties. Thus the weathering of coal after exploitation presents the same phenomena as when coal in its native bed has been exposed to superficial agencies by erosion. And all coal within moderate depths from the surface has ever since its deposition undergone secular alteration varying in degree with many widely different circumstances.

Soluble salts resulting from the epigenesis of sedimentary portions

of coal-beds are often diffused or eliminated, and therefore seldom manifest themselves. When, on the other hand, chemical action is accelerated by direct exposure to superficial agencies, some of its effects become visible. The impregnation with mineral salts of water flowing from coal seams deep under cover, tends to show that chemical reactions are not confined to coal within close proximity to the surface, however slower or more feeble these reactions may be the deeper they take place.

Among the more common products of epigenesis in coal seams are especially salts of iron, alumina, lime, and magnesia, including besides those already named ferrous sulphate, or sulphate of protoxide of iron (melanterite, or green copperas), and ferric sulphate (coquimbite, or white copperas). Among the efflorescences of highly weathered coals, such as may be seen in the excavations of old mines, other hydrous sulphates of iron may probably be recognized. Hydrous sulphates of alumina are also found under the same circumstances, such as halotrichite, alunogen and löwigite. Other alums found in bituminous schists can hardly be absent from the richly aluminic and styptic efflorescences of earthy and pyritous partings of highly weathered coals. Double salts of iron and alumina, or of iron and magnesia, of the same series of sulphates as the aluminio-ferric sulphate observed by Fleck and Hartig,* suggest their presence under the supposed circumstances, which at least would be favorable to the formation of such compounds. A difficulty to exact identification of the separate salts entering into the composition of such efflorescences arises from their multiplicity, and ordinarily the minuteness of their crystals, which prevents their isolation.

A sample of cretaceous anthracite brought from Peru by Mr. H. Bauerman, yielded by analysis, according to Dr. Percy, 10.35 per cent. of sulphur, and only 3.75 per cent. of ash. Of this excessive proportion of sulphur only 3.52 per cent. was evolved on heating the coal for an hour without access of air to an intense white heat. The ash consisted principally of silicate of alumina, and contained only 0.018 of sulphur. The retention by the carbonaceous residue of so large a proportion of sulphur in excess of any possible combination in the ash, points, as suggested by Dr. Percy, to the combination of by far the larger part of the sulphur in this coal with its organic constituents.† Another example, leading to the same in-

* Technik der Steinkohlen, ii, 219; Dana's Min., 650.

† Percy's Fuel, Appendix, p. 567.

ference, is given by Dr. Percy. A miocene caking coal from New Zealand yielded 2.50 per cent. of sulphur against 3.50 per cent. of a white ash, and proved to be free from any sulphate soluble in hydrochloric acid. Hence the suggestion that the sulphur may exist in the coal in the same state as that in which it exists in albumen, fibrin, hair, etc. Percy has also suggested the inquiry whether the assumed organic combination of sulphur in examples of coal like the above may not be referable to the presence of some resiniferous compounds like tasmanite of the resiniferous or combustible shale described by Professor A. H. Church, the notable proportion of sulphur in which is shown to be in combination with the carbon and hydrogen of this substance, of which the following is the centesimal composition, apart from the ash, which is mainly an inseparable mechanical mixture of silica and alumina from the matrix of the mineral, viz.: carbon, 79.34; hydrogen, 10.41; sulphur, 5.32; oxygen by diff., 4.93. That the large amount of sulphur is an integral part of the carbonaceous matter itself, and was not owing to the presence of an inorganic sulphide or sulphate, was proved in several ways, and was further confirmed by the observation that the more completely the mineral matter had been removed the more sulphur was found. In the series of analyses given by Church several solvents and oxidizing agents were employed. A certain amount of sulphur was found in the ash, with ferric oxide as a soluble sulphate, from the oxidation, it is supposed, of the sulphur of the tasmanite proper, but it was necessary to completely destroy and dissolve the mineral in order to extract the whole of the sulphur.*

The artificial compound of sulphur, carbon, and hydrogen in vulcanite, or vulcanized caoutchouc affords a better illustration of the existence of sulphur in organic combination with hydrocarbons. Caoutchouc is composed wholly, according to separate experiments of Ure and Faraday, of carbon and hydrogen, containing 87.5 per cent. of carbon, and 12.5 of hydrogen. In the process of vulcanization a small proportion of the sulphur appears to combine directly with the rubber. Dr. Watts gives this proportion as one- or two-hundredths of the weight of the rubber.† A much larger proportion remains interposed between the pores, and may be extracted by the action of solvents, friction, or alternate extension and contraction. The total change in the physical properties of vulcanized rubber,

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* Philosophical Magazine, vol. xxviii, 469.
† Watts's Dictionary, i, 739.

together with the fact of its insolubility in the ordinary solvents for rubber, shows the nature of this artificial compound to be chemical, and not mechanical. According to Dr. C. F. Chandler, no appreciable quantity of hydric sulphide is evolved in the operation of vulcanizing at a high temperature; hence it is improbable that a substitution of sulphur for hydrogen occurs.*

The vulcanization of gutta and other hydrocarbons, like certain oils, and their conversion into hard substances by the chemical action of sulphur and certain sulphides is well known.

Upon digesting a variety of plants at a temperature of 167° to 212° F., for two years, Göppert obtained a product which could not be distinguished by the eye from brown coal, but which acquired the black, shining aspect of mineral coal only after the addition of a small quantity of ferrous sulphate ($\frac{1}{18}$).† This was of course reduced to ferrous sulphide.

The relative amount of sulphur found by Mr. McCreath in a number of bituminous coals from different parts of Pennsylvania varied from 0.425 to 8.427 per cent. According to this chemist, the mean average of 34 coals from Clearfield County gave 1.36 per cent. of sulphur; that of five specimens from Centre County 0.767 per cent.; that of 37 specimens from Jefferson County 1.518 per cent. Averages of Armstrong County and Clarion County coals gave 1.57 and 3.30 per cent. of sulphur respectively.‡

The average proportion of sulphur present in a number of bituminous coals of Ohio, recently examined by Dr. Wormley, was 1.551 per cent.; that of the coals from the lower (southern?) half of the State being 1.229 per cent., and that of the coals from the upper (northern?) half, 1.836 per cent.§

Mr. I. Lowthian Bell gives the mean average of sulphur contained in well-known British coals as follows:||

18 Newcastle,	1.24
36 Wales,	1.43
8 Scotland,	1.11
7 Derbyshire,	1.01

Interesting and useful as experiments in the laboratory often are

* Chandler, in Johnson's Cyclopædia, ii, 1170.
† Jour. Geol. Soc., i, 33.
‡ See Geological Survey of Pennsylvania, Report M, p. 30.
§ Geological Survey of Ohio, Annual Report for 1870, p. 411.
|| Rep. Brit. Ass. Adv. Sci., xxiii, p. 742.

as to the composition and physical properties of coke, care must be taken to distinguish between coke so made and such as is made on a working scale. For oftener than not wide differences, both chemical and physical, will be found between cokes prepared from a single quality of coal by the two methods, or even by different processes on an equal scale. Especially as to the degree of the volatilization and oxidation of sulphur by combustion do the thorough and painstaking manipulations employed in the analysis of coal and of crucible-made coke differ from related industrial operations, even though identical in principle. The desulphurization of coke may be, and generally is, carried farther in the laboratory than is practicable on a working scale. The common omission to distinguish between *artificial* or crucible-made coke and *industrial* coke, so called by Philippart, is too often the cause of disappointment in practice, not only because laboratory-made coke, for obvious reasons, is almost always lower in sulphur than coke made in ovens or piles, but because it often further varies from the latter in other essential chemical and physical properties. The desulphurization of even crucible-made coke seldom reaches the point of theoretical desulphurization. The mechanical and thermal difficulties opposed to the free chemical action required for complete desulphurization, are of course far greater in the case of coking in large masses, and, as always happens, except when coal is washed, without pulverization.

In an examination of crucible-made cokes from Ohio coals, Professor Wormley found them to differ greatly as to the relative proportion of sulphur eliminated and that retained in the coke. Thus specimens from the lower and second layer of the Straitsville or Nelsonville seam of coal in the Hocking Valley district of Ohio, containing respectively 0.49 and 0.93 per cent. of sulphur, lost in coking (in the crucible) as high as 0.41 and 0.90 respectively. On the other hand, a number of analyses of coals from the same seam show the cokes to retain a large proportion of the amount of sulphur originally in the coal. The average loss of sulphur in coking six samples of Straitsville coal was no more than 56 per cent.*

Of the fourteen Pennsylvania bituminous coals examined by Mr. McCreath† for sulphur after artificial coking, nearly all were found

* Geological Survey of Ohio, Annual Report, 1870, p. 411; Annual Report for 1869, p. 108.

† In a "Table showing the percentage of iron and sulphur in different coals, and the proportion of the sulphur which is volatilized by coking, etc.," given in Report MM of the Second Geological Survey of Pennsylvania, and prepared

to retain a large proportion in the coke, the loss in coking in no case exceeding two-thirds of the amount originally present in the coal.*

Mr. Bell gives the percentage of sulphur in the coke used at Cleveland and made from South Durham coal as 0.44 to 0.77, or an average of about 0.60 per cent.†

Determinations of sulphur in coal or coke are wanting in thoroughness, and fall far short of intelligent application to technical purposes, when not carried to the point of distinguishing the several states of combination of this substance. Not only does the relative elimination and retention of sulphur in coke usually depend in some degree upon the relative proportion of sulphides and sulphates, but the relative proportion of hurtful sulphur and harmless sulphur depends upon the same relation. This relation may sometimes be altered without difficulty, so as to become more favorable for certain purposes. Sulphur in coal can hardly be regarded as fully determined unless rendered as follows:

Sulphur in the state
- Of bisulphide.
- Of soluble sulphate (readily soluble in water).
- Of sulphate (soluble in 400 to 500 parts of water).

In coke the determination of sulphur should be made as follows:

Sulphur in the state
- Of bisulphide of iron.
- Of protosulphide (and subsulphide) of iron.
- Of sulphate.

Such determinations are, to be sure, reached not without more labor than is required by the prevailing incomplete analytical prac-

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by the chemist of the Survey from twenty-five recent analyses of coals and cokes from that State, it appears that the proportion of sulphur expelled by artificial coking varies from 14 75 to 57.92 per cent. of the whole amount of sulphur present in the coal, the average being 38.50 per cent.. Seven of the number of coals, containing 63.51 per cent. of their total sulphur in excess of the requirements of the iron found, to form bisulphide, lost 34.57 per cent. of the total sulphur by coking in the crucible. On the other hand it is shown that eleven of this series of coals, with an average of only 11.36 per cent. of the sulphur not combined with iron in the form of pyrites, lost 37.88 per cent of the total sulphur. Again, two coals, with an average of 74.58 per cent. of sulphur in excess of the amount present in the pyrites, lost 20 97 per cent., while two other coals, with only 2.20 per cent. of a similar excess of sulphur, lost 44.81 per cent The average proportion of sulphur in the whole series of coals was found to be 2 138 per cent., and in the cokes made therefrom, 1.912 per cent. (With reference to these results see a previous footnote to this paper, page 185.)

* Geological Survey of Pennsylvania, op. cit., p. 31.

† Brit. Ass. Adv. Sci., op. cit.

tice of at once determining all the sulphur as sulphate. The coal and its products should be treated in separate parts by different solvents, as remarked by Dana, and no analysis of coal can be regarded as satisfactory where this is not done.*

It is no part of the purpose of the present paper to treat of the several improved methods of analysis proposed by such authorities as Calvert,† Eschka,‡ and Stock,§ providing for discriminating and searching analysis of sulphur in coal and coke, in place of the present superficial practice, which is retained rather by habit, or as a time-honored custom, but which certainly is far behind the requirements of the day. Percy has pointed out several sources of error in the chemical and stochiometrical methods, as commonly practiced, of determining sulphur in coal and coke.

In coal under destructive combustion with free access of air, a portion of the sulphur of the pyrites is oxidized and volatilized as sulphurous acid and sulphurous anhydride, and the remainder still further oxidized to the state of sulphuric acid, which, in the presence of lime and magnesia, forms sulphates of these bases undecomposable at a red heat. Ferrous sulphate formed at the less elevated temperatures of combustion becomes further oxidized at an increased temperature to ferric sulphate. As combustion becomes more intense oxidation of carbon continues at the expense of one or both of these oxygen salts. Ferric sulphate becomes reduced successively back to ferrous sulphate and thence to protosulphide of iron, and so on until all the sulphur is expended, and the residue left as ferric oxide. The reduction of ferrous sulphate without intermediate further oxidation doubtless happens to some extent.

Bisulphide of iron (pyrites) heated in contact with carbon parts, according to Philippart, with one-half of its sulphur in the state of bisulphide of carbon, and is completely converted into protosulphide. At a still higher temperature this residue further parts with a small proportion of sulphur, when its composition is found to be that of a subsulphide corresponding with the formula $Fe_2S+6FeS$. The total loss of sulphur after this formula is $\frac{9}{16}$, or 56 per cent.||

Bisulphide of iron is converted at a high temperature into protosulphide of iron, which remains unaffected by a higher temperature, or when heated in hydrogen. Exposed to vapor at a high temperature it decomposes rapidly, liberating hydrogen, and hydric sulphide,

* Dana's Mineralogy, p. 756.

† Chem. News, 1871, xxiv, 76.

‡ Jahresbericht, 1875, xx, 1020.

§ Ibid., 1874, xxx, 1211.

|| Revue Univ., op. cit. 274.

leaving a black residue which is attracted by the magnet. Thus, according to Philippart, Regnault found in operating on four grams in a glass tube, that one-half of the sulphur had disappeared at the end of three hours.*

Although bisulphide of iron at a high temperature and with free access of air is converted into protosulphide, this reaction in the process of coking at an equally high temperature cannot be completed without such access of air and such a duration of the process of calcination as to excessively oxidize the hydrocarbon of the coal; or, in other words, as will suffice for destructive combustion. Philippart's experiments with industrial coke to ascertain the maximum desulphurization compatible with the least oxidation of carbon resulted in a loss of only 10 per cent. of sulphur, attended with a loss of carbon amounting to 55 per cent. In crucible-made coke better results were obtained, the loss of sulphur amounting to 72.50 per cent., attended by a loss of carbon of 27.50 per cent., ten hours having been given to the operation. In another experiment lasting twenty hours, 30 per cent. of the total sulphur was converted into sulphurous acid and anhydride, and 18 per cent. of carbon lost by oxidation. In the first experiment 0.08 per cent. of the original sulphur existed in the form of sulphate, and in the second 0.05 per cent. After the experiments the sulphates were found to be respectively 0.70 and 0.125 per cent. In the experiment with industrial coke the sulphates were increased from 0.075 to 0.550 per cent.†

I shall presently refer to the belief that for smelting, as well as for gas-making, sulphur in the form of sulphate is, if not innocuous, its least objectionable mode of occurrence in coke and coal. In any process of coking, therefore, with the combustion limited in time, and also limited as to access of air, only a part of the bisulphide of iron passes over to the state of protosulphide. Thus Philippart found that the mean relative amounts of bisulphide and protosulphide of iron retained in industrial coke corresponded to the formula $FeS+3FeS_2$, or to a loss of sulphur of 14 per cent.; that is, to one-fourth of the loss as compared with theoretical desulphurization. In experiments on five samples of crucible-made coke the same authority found the loss of sulphur to vary from 11 to 56 per cent.; in one case only in five reaching the point of complete desulphurization.‡

* Revue Univ., op. cit. 274.

† See Percy's Fuel, 470.

‡ Philippart, op. cit. 275.

The objection is made by Percy against Philippart's conclusion that the temperature of an ordinary coke oven is sufficient to wholly reduce bisulphide of iron to protosulphide, and further eliminate sulphur as bisulphide of carbon. But the small loss of sulphur observed in the very full series of analyses of industrial coke reported by this chemist, and also generally in analyses of oven-made coke, leads, it seems to me, to no other conclusion. Unexpected as such results may be when compared with theoretical requirements, the disparity is certainly no greater than that which is observed in the relations of sulphur between industrial coke, and the carefully manipulated product of the laboratory, where every requisite condition is under control. The explanation of the incomplete reduction of bisulphide of iron in coking on a large scale is to be found (1) in the limitation of the time given to the operation, and (2) in the impenetrability of particles of pyrites to oxidizing agencies, especially in the more thoroughly caking coals, the tendency of which is to envelop a part of the pyrites in an impervious coating, and by dense agglomeration to exclude the air from very considerable portions of the coke mass.

The same physical difficulty extends to the several methods proposed for the desulphurization of coke based upon the principles of oxidation, such as the employment of currents of air or of compressed air, and suffices to account for their failure. The same remark equally applies to desulphurization by hydrogen from the decomposition of steam.

But for such irregularities in coking the diminution of sulphur would be in some more constant proportion to the element of time given to the operation and to the degree of oxidation of carbon. That the physical conditions are far from uniform even in a single oven in respect to the whole mass of coke, to say nothing of ovens of different construction nor of different varieties of bituminous coal, is observed wherever the operation of coking is conducted on an industrial scale. The mass of coke which at first agglutinates at length intumesces, and finally becomes fissured by the expansive force of gases. While surfaces are thus presented to oxidizing agencies, the interior portions of intermediate masses, thus separated by fissures, are comparatively inaccessible to the action of air or steam, notwithstanding the increasing vesicular state of coke as the operation continues, filled as the cells must be by reducing gases.

In the incomplete and uneven combustion of coal in coking on an industrial scale by ordinary methods, at a general elevation of tem-

perature not much above red heat, and with only a limited and locally variable (and in fact fluctuating) access of air, more or less, but not all, of the bisulphide of iron passes into the state of protosulphide by the loss of half of its sulphur. Through inequality of the physical conditions above referred to, as well as of the temperature at which combustion takes place, the formation of the lower sulphide (subsulphide) observed by Philippart, is not improbable as indicated by Percy.* The sulphur set free in part escapes as sulphurous acid or sulphurous anhydride, in part as hydric sulphide, and probably another part as bisulphide of carbon, from at least such portions of the mass as have been exposed to a sufficiently high temperature, as must sometimes happen through inequality of temperature. The remaining portion of sulphur further oxidized to sulphuric acid forms ferrous sulphate, passing, as already detailed, by continued oxidation at a higher temperature into ferric sulphate, and, as under other circumstances, in the presence of lime and magnesia, sulphates of these bases. Notable proportions of calcic sulphate appear to be present in the coke from almost all coals, according to available analyses, although sulphide of calcium may sometimes be the intermediate or ultimate form of the combination in coke, through reduction of the sulphate or by the decomposition of protosulphide of iron in the presence of this alkaline earth.

Attempts to overcome the resistance of coke to the penetration of air by compressing it two to two and a half atmospheres have signally failed in France and Belgium to effect desulphurization to a greater degree than 12.50 per cent. in one case, and in the other 7 per cent. of sulphur originally existing in the state of sulphide.† Elevation of temperature caused by increased pressure of air must result in a larger consumption of carbon.

The gases resulting from the decomposition of steam by incandescent coke unite with the gaseous products of combustion, forming, in the order of their volume, hydrogen, carbonic oxide, carbonic acid, marsh gas, and hydric sulphide—the last from the decomposition of protosulphide of iron.

Sheerer's experiments‡ with a high pressure of steam at the end of the operation of coking are supported by analyses too few in number to be conclusive, but they tend to show that an important degree of desulphurization may be attained by the action of steam, though, as suggested by Percy, not without considerable loss of carbon above

* Fuel, p. 469. † Revue Univ., p. 288.

‡ Berg u. Hüttenmännische Zeitung, xiii, 239; Percy, Fuel, 468.

what is essential to the operation of coking. The question therefore arises whether the same loss would follow from the application of steam earlier in the process, or whether the requirements of gaseous combinations would not be met by the ordinary gases of combustion without further loss of carbon after the operation is complete. The physical difficulties above referred to, in every process of coking, as well as the uneven distribution of pyrites in the charge, require the confirmation of Sheerer's experiments by repeated analyses.

The odor of hydric sulphide from coke when extinguished with water can only proceed, as remarked by Percy, from very small proportions of this gas, notwithstanding its powerful effect upon the organs of smell. By the contraction and disintegration of coke, under the action of a stream of water, sulphurous anhydride and other gases occluded within its interior portions, add doubtless to the pungent odor which it emits when treated in this manner.

The application of several reactions first pointed out by Berthier has been suggested by different authorities for the desulphurization of coke, or rather for the transformation of its sulphur from its more readily decomposable forms into other combinations capable of more effectually resisting decomposition under the influences to which it may be exposed, especially in the blast furnace.

Protosulphide of iron at a red heat in contact with carbon, and intermixed with once or twice its weight of an alkaline carbonate (potash or soda) or an alkaline earth (lime or baryta), is decomposed; metallic iron is separated and a double sulphide formed of iron and potassium, sodium, calcium or barium, as the case may be.*

Similar reactions are attributed by Philippart to an admixture with alkaline chlorides, resulting when salt is used in chloride of iron and sulphide of sodium. Calvert's process of desulphurization of coke seems to be based upon the action of steam upon these products of the decomposition of protosulphide of iron. At a very high temperature and in the presence of carbon it would appear *a priori* that a great part of the alkaline metal becomes volatilized before it can have completely reacted on the protosulphide of iron, while it is probable that a considerable portion of the pyrites remains unchanged in all coke ovens.

Of the number of alkaline products above named, soda salts and lime are alone afforded at such a cost as to be available for such a purpose. In coke made on an industrial scale with the use of these

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* Berthier, Traité, ii, 192.

salts separately, Philippart found the quantity of sulphur in the state of sulphate not sensibly diminished, and therefore concluded that they admit of no advantageous application in practice, however different the result obtained in the laboratory. Calvert's method tested on the same scale, while found to be more effective than the use of steam alone, resulted in too little benefit to warrant its cost.*

Upon making a number of crucible cokes with different proportions of lime, Philippart found their tenor of sulphur in the state of sulphide of calcium, within the limit of the experiments, to be in proportion to the quantity of lime employed. A portion of the lime combined with the silicate of alumina corrresponding to the slate contained in the coal. In withdrawing the coke from the crucible while still hot no odor of sulphurous acid was perceived, thus proving the complete combination of the remaining sulphur with calcium. The same fact was proved by the increase of sulphur in the calcareous coke as compared with that made by the ordinary method. The coke was pronounced sufficiently compact, though wanting somewhat in tenacity. Fusions of iron ore in a brasqued crucible with calcareous coke, made with the use of 12½ per cent. of lime, established the complete innocuousness of the sulphide of calcium, not only by the quality of the metal produced, but also by the fact that this sulphide was found with the slag. The conclusion was thus reached, that an intimate intermixture of coal and carbonate of lime in pulverized or washed coal tends to overcome the injurious effects of the sulphur corresponding to the original sulphide of iron.†

Berthier determined that at an elevated temperature in the presence of carbon the basic alkaline silicates and borates, and the basic silicate of lime, free or combined with basic silicate of alumina, decompose protosulphide of iron to a degree proportionate to the elevation of temperature, by the action of the alkaline base above considered reduced from the state of silicate, the excess of base in the original silicate still remaining combined with the acid. This reaction is observed in blast furnaces using coal or coke with more or less aluminous ores, the slags of which contain sulphide of calcium.‡ So, too, the reaction between sulphide of iron and the alkaline earth, lime, and its carbonate is supported by the common practice of the blast furnace for the diversion of sulphur from the metal into the slag, by the use of an excess of lime within the limits consistent with the requisite fusibility of the slag.

* Revue Univ., op. cit., 474; Percy, Iron and Steel, p. 36; Fuel, p. 474.

† Revue Univ., op. cit., 306.

‡ Berthier, Traité, ii, 193.

In the manufacture of foundry iron at establishments where the coal is washed in a pulverized state previous to coking, the liming of coke would be attended with an additional cost, but small in comparison with the advantages which in some cases might be expected from Philippart's conclusions. That the simple experiment has not been tried, so far as I am aware, at such works producing iron with an excess of sulphur referable to the coke, is somewhat remarkable. It is true, however, that the present mechanical appliances for the purification of coal are very effective in the case of many coals for the removal of pyrites. But the efficiency of the washing process depends equally upon the coal, especially upon the distribution of the pyrites, its size, and its relation to the lighter or more bituminous portion of the coal. In the case of coal where the pyrites is not concentrated with the heavier or slaty laminæ, but on the contrary is disseminated in thin scales adhesive to the bituminous, and therefore the best, portions of the coal, it is probable that the supplementary liming might result in no little advantage.

In estimating the value of any process for the manufacture of calcareous coke on the principle of the stronger affinity of sulphur for calcium than for iron, the question arises whether the same effect is not to be attained in the blast furnace with no greater consumption of lime, especially as by the escape of a larger proportion of sulphur by the ordinary methods of coking, less remains in the coke to be transformed into sulphide of calcium. The gain, if any, of the previous transformation of sulphur, would be in the fact that the pulverulent condition of the lime and its intimate admixture with the coal must necessarily facilitate the reaction, no such close contact between particles of lime and fuel taking place in the blast furnace.

It has long been observed that manganesiferous ores in the blast furnace afford slags rich in sulphur and iron proportionately low in this substance.*

According to Berthier, silicates or borates of manganese behave in the same manner towards protosulphide of iron as the alkaline silicates and borates.†

The calcination of pulverized coal with peroxide of manganese in different proportions produces a coke, according to Philippart, in which a large proportion of the sulphur originally present in the coal is fixed as manganous sulphide. In the blast furnace a small proportion of manganese and perhaps a corresponding amount of

* Wedding's Percy's Eisenhüttenkunde, p. 49.

† Traité, p. 193.

sulphur passes into the metal. As a means of the transformation of sulphur in coke the action of manganese is like that of lime, but according to this chemist, less efficacious.*

Tunner asserts that the diversion of sulphur from pig iron to the slag, by the use of manganese in the blast-furnace, is *more* effective than by the use of lime, and that however difficult to reduce, it is less so than lime. To the determination of the sulphide of manganese to the slag, Tunner ascribes the excellence of the iron made in the coke furnaces at Prävali (Carinthia) and Schwechat (near Vienna), in spite of the constant presence of sulphur in the coke rising from three per cent.†

The sulphates of calcium and magnesium melt at a high temperature without decomposition, but in the presence of carbon are reduced to sulphides. This reaction doubtless takes place in the blast furnace, and to some extent also in the coke oven, depending under both circumstances on the degree of elevation of temperature and predominance of reducing or oxidizing conditions as the case may be.

Hence it follows that in estimating the amount of sulphur in coals and cokes it is well to distinguish the proportion of sulphate in combination with lime and magnesia, for while swelling the aggregate of registered sulphur it may be regarded as comparatively innocuous both for the blast furnace and for gas-making.

At ordinary temperatures, under exposure to the atmosphere, bisulphide of iron in the form of white pyrites or marcasite readily decomposes, but far more slowly when in the form of yellow pyrites or pyrite. Such a difference as to facility of oxidation appears to be due to molecular differences arising from crystalline structure. Marcasite occurs in several states of crystallization and aggregation favorable to oxidizing influences as compared with the usually more compact mode of occurrence of pyrite. It is in the more readily decomposable form of marcasite that bisulphide of iron generally occurs in bituminous coals and lignite, while in anthracite it is generally, if not always, present in the form of its more stable species. By atmospheric oxidation at ordinary temperatures bisulphide of iron in either form is oxidized to ferrous sulphate and ferric sulphate successively, probably with the separation of minute proportions of free sulphur. These salts are frequently found accompanied as above remarked by aluminic sulphates (alums), the result of the decomposition of the more aluminous partings of coal-

* Revue Univ., op. cit., 311.

† Jahrbuch d. Bergak. zu Leoben, xxii, 184.

beds in which pyrites is generally more or less concentrated. These several salts, together with the sulphur, appear as a whitish efflorescence, coating the surfaces of weathered coal, and sometimes accompanied by calcic sulphate. As these salts, with the exception of calcic sulphate, are readily soluble, they may be recognized by the taste characteristic of salts of iron, or by the bitterness characteristic of alum. As even gypsum is soluble in 400 to 500 parts of water, all of the salts may sometimes disappear from the surfaces of coal under the action of rain or in the process of coal-washing. A reaction between salts of iron in weathered coal and the calcareous waters which freely circulate within coal-bearing strata, results in the deposition of ferric oxide and gypsum. The former gives rise to the rusty aspect of coal weathered *in situ*. According to Richters, ferric oxide is capable of reduction to ferrous oxide by contact with coal. Thus by alternate oxidation and reduction it may play an important part as a carrier of oxygen to the coal.*

While the ratios of sulphur in coal may not in some cases be sensibly decreased by its transformation into sulphates through weathering, especially when these be not subsequently eliminated, the result of this process in coal, so far as it affects this substance alone, is an important amelioration of the fuel, especially for the blast furnace. The benefit is far greater if the soluble products of oxidation are removed by exposure to rain, or by water purposely applied, and sometimes indeed may be deemed sufficient to offset the loss in calorific and physical conditions which coal always suffers in weathering. Sulphates of iron, although decomposable at elevated temperatures, and reducible in the presence of carbon, are but an inconsiderable source of sulphur in pig iron, as each successive change of its state is attended with loss of sulphur itself, while basic slags readily take up and retain such sulphates so far as they remain undecomposed. The same may be said of the less decomposable but equally reducible calcic sulphate. Hence, again, the importance of discriminating between sulphides and sulphates in the determination of sulphur in coal, especially when it has been weathered. The most thoroughly weathered coal will be found to be lowest in sulphur in the state of sulphide, however high in comparatively harmless sulphates when not removed by the action of water.

In the dry distillation of bituminous coal, as in the manufacture of illuminating gas, the temperature employed, which is between a

* Dingler's Polytech. Jour., cxcv, 194.

red and white heat, is only sufficiently elevated to convert bisulphide of iron into protosulphide. This change is attended with the evolution of hydric sulphide and sulphurous acid, probably passing in the last stages of the operation into sulphurous anhydride, and with the formation of bisulphide of carbon vapor in about the same stages of the operation. The protosulphide of iron remains in the coke without further decomposition. Among the volatile products of the distillation of coal, namely, the gas, ammonia water, and tar, other sulphurous combinations are recognized, especially sulphocyanide (NH_4CNS) and hydrosulphate (NH_4HS) of ammonium, probably oxysulphide of carbon (CSO), and according to Hofmann, hydrosulphocyanic acid ($HCNS$).* All of these compounds produce sulphurous acid when the gas is burned, and probably some sulphuric acid, which forms with the liberated ammonia the sulphate of that base. The purification of gas, having for its object the removal of carbonic acid and these sulphur compounds, is more or less perfectly effected by dry lime, clay, or ferric oxide, which combine with sulphuric acid, absorb carbon, and with the aid of the hydrogen in the gas decompose the volatile sulphides, especially the sulphides of carbon and hydrogen.†

ATMOSPHERIC OXIDATION OR WEATHERING OF COAL.

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By the term *weathering* of coal is meant the process of deterioration to which under various circumstances it may be exposed at ordinary temperatures, both from outward agencies on the one hand, and on the other hand from its own constitutional qualities. This process involves along with some minor qualitative changes certain quantitative changes resulting in a gradual and limited reduction in the quantity of the organic elements, and relatively a corresponding increase of its inorganic or mineral constituents as represented by its ashes, and often in an increase of the absolute weight of the coal.

In common with many mineral substances, coal absorbs oxygen

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* Percy's Fuel, 419.

† Watts, Dictionary of Chemistry, i, 776, 1038; C. F. Chandler, American Chemist for March, 1874.

from the atmosphere through the chemical affinity for oxygen possessed by its elementary components. One portion of the oxygen absorbed combines with a portion of the carbon of the coal and forms carbonic acid; another portion with the disposable hydrogen with the formation of water; and a third portion with organic parts of the coal, entering into unknown combinations, while the remainder is consumed in oxidizing the iron and sulphur of bisulphide of iron (iron pyrites), the latter to the state of sulphuric acid, which unites with any bases present, such as iron, lime, alumina or magnesia, in the form of more or less soluble sulphates. When preserved from the dissolving action of water, these appear as an efflorescence on surfaces of the coal. As such changes are attended by changes of volume they generally, to a greater or less degree, result in mechanical disintegration.

The practical effect of the weathering of coal, while sometimes increasing its absolute weight, is to diminish the quantity of carbon and of disposable hydrogen, and to increase the quantity of oxygen and of indisposable hydrogen. Thus the weathering of coal involves a loss of calorific carbon and hydrogen, and a corresponding gain of noncalorific carbonic acid and water. Hence a reduction in the calorific power of the coal, and a reduction accordingly of its value.

The weathering oxidation of coal is the result of separate but closely related phenomena, the effects of which it may be well to here distinguish as follows:

1. The invisible effect of the oxidation of its organic constituents.
2. The more or less visible effect of the oxidation of its accessory pyritic contents.

The published results of the first special investigation on a large scale of the effect on coal of exposure to the weather, namely, that by Professor Grundmann, appearing in 1862, were indeed startling and even alarming. By comparative estimation of the ashes of a Silesian coal (Königshütte) before and after exposure to all conditions of the weather in a stock pile of small coal containing some 300 tons, that chemist announced a loss in volatile matter of not less than 58.2 per cent. in nine months. Of this enormous percentage of loss only 1 per cent. corresponded with the loss in moisture, against a total of 5 per cent. of moisture in the fresh coal; while 58 per cent. was referred to the first five of the nine months. A second sample, immediately protected from falling rain and snow, sustained a loss of not less than 43 per cent. in seven months. Both samples lost their power of coking in two to three months. While

the specific gravity of the coal was stated not to have altered, nor the hygroscopic water to have increased but slightly, the proportion of ash had relatively increased according to the loss above noted, and the centesimal composition of the coal, exclusive of the ash, appeared to preserve its original relations.*

Important and valuable a contribution to the knowledge of the subject as Grundmann's researches undeniably are, they have failed to be confirmed by subsequent experiments which they were well calculated to suggest and stimulate. Reder, Richters and others have pointed out sources of error in some of Grundmann's experiments, and shown some of his conclusions to be unsupported, not only by experiments of their own, but to be hardly compatible even with certain physical results obtained by himself. Notwithstanding the obvious exceptions to these conclusions especially in the light of subsequent investigations, and the general fact that the comparative estimation of the ashes of coals affords no proper basis for the investigation of this question, without precautions against lack of uniformity in samples, Grundmann's results have of late been repeatedly quoted as affording examples, albeit extreme examples, of the weathering of mineral coal.

By another computation from a portion of Grundmann's data the reduction in calorific power of the Königshütte coal does not exceed 11 per cent. Reder, however, after exposing to weathering action the same coal, together with English Brancepeth, and Hanoverian (Osnabrück) coals, found that at the end of a year all had gained rather than lost in weight, that the ash of neither had increased, and that the Silesian coal alone suffered any reduction of coking power, of which some remained after four months.

Some of these discrepancies were subsequently explained by Grundmann,† and others are doubtless to be referred to lack of uniformity in the samples employed by both experimenters, while the different methods adopted can have led in either case only to approximate results. Similar conclusions followed from like inexact experiments instituted by the management of the Hanoverian Railway, viz., besides that of Reder at Osnabrück, others at Harburg with English coal, and at Hanover with Shaumburger smith-coal. No practical deterioration of the coal was observed in either case. In coal from the Glücksburger seam at Ibbenbüren, however, Reder observed a

* Carnall's Ztsch., X, 1862, 236.

† Oest. Ztsch. f. Berg, u. Hütt., XV, 1867; 270. Kerperly Bericht, 1866, p. 33.

loss of calorific power at the end of a year amounting in one case to 6 per cent. and in the other to 2.6 per cent., and also a loss of coking power in the same coals of 4.6 and 2.1 per cent, respectively.*

Grundmann's conclusions received partial confirmation from Varrentrapp's experiments, first upon lignite and afterwards upon Westphalian gas-coals.† The latter were observed to part with carbonic acid at ordinary and at elevated temperatures in quantity nearly proportionate to the elevation of temperature, more than one-half of the carbon having been converted into carbonic acid in the space of one hundred days at a temperature of 150°–160° C.

Fleck examined in 1865 a series of six Saxon coals which had been preserved in a cabinet since 1856. The results of his examination were then compared with analyses made nine years previous, when, it is presumed, the coals were in a fresh state. After such an exposure in a dry place, three of the number, it was inferred, showed an important increase in the proportion of ashes together with a corresponding decrease of organic matter, while the ashes of the remaining three appeared to have decreased. The explanation of such unlike results was sought by Fleck in the lack of uniformity of the related samples, especially as to the ratio of ashes. An increase of oxygen. and of indisposable hydrogen was observed in each case, together with a loss of carbon and of disposable hydrogen. Hence the conclusion, that at ordinary temperatures bituminous coal sustains a loss in carbon and in disposable hydrogen, and proportionately a loss in calorific value, which is increased in proportion to the addition of combined water.‡

Fleck was thus the first to indicate the nature of the chemical changes in coal from weathering, although the conditions of his experiments did not admit of an estimation of their practical effect. Especially was the quantitative estimation of the degree of these changes vitiated by an uncertainty as to the uniformity of corresponding samples, rendered all the greater by an excessive loss of combustible matter in some instances, as shown by the increase of ashes. Nor does it appear whether the loss in carbon and hydrogen was due to a gain or loss in the absolute weight of the coal, this point, of course, it having been impracticable to determine.

Every chemist has observed the oscillations in weight of mineral coal of the bituminous class at elevated temperatures, especially in

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* Oest. Ztsch. op. cit. Reder, Ztsch d. Ver. deutsch. Eng. X, 698.

† Dingler's Jour., vol. 175, p. 156; vol 178, p. 379.

‡ Technik d. Steinkohlen Deutschlands, ii, 221.

the drying of coal preliminary to analysis. Such coals in a dry state at a temperature of 180° to 200° C., are often found to increase in weight, the maximum of which is finally reached when maintained at a constant temperature. This phenomenon was investigated by Richters in 1868, and by him considered to be due to the chemical absorption of oxygen at the same time that hydrogen is evolved, and hygroscopic water evaporated.* Upon a closer investigation of this phenomenon, Richters subsequently found that in proportion to the absorption of oxygen, carbonic acid and water from the oxidation of carbon and hydrogen are expelled; and that all such action ceases with the increase of weight, or, conversely, that the increase of weight ceases with the oxidation of carbon or hydrogen. Hence it was inferred that the amount of oxygen absorbed is in definite proportion to a certain readily oxidizable part of the carbon of the coal, and to a certain readily oxidizable part of its hydrogen—namely, the disposable hydrogen, so called. This conclusion is in agreement with that view of the composition of mineral coal which regards its carbon to be present in the two separate states, of pure carbon on the one hand, and, on the other hand, carbon in organic combination, the precise nature of which is not yet fully understood. The evolution of carbonic acid, together with the absorption of oxygen, is found to be greatest at first. These facts bear out the theory of the existence of carbon in the two states above mentioned, the combined carbon only yielding to oxidation at higher temperatures than those apparently requisite for the oxidation of carbon in an uncombined state. Further, the probability that the absorption of oxygen is in proportion to the disposable hydrogen, amounts to almost a certainty, as suggested by Richters, when considered in relation to the phenomena of oxidation displayed by wood.

Saussure observed that 240 parts of dry oak-wood shavings, while sustaining a loss of 15 parts in weight, converted 10 cubic inches of oxygen, without change of volume, completely into carbonic acid, containing 3 parts of carbon by weight; 12 parts of water from the elementary substance of the wood were, therefore, separated. So, too, Liebig remarked that shavings of green wood at first reduce the volume of oxygen, while seasoned wood in a damp state converts oxygen into carbonic acid without change of its volume.† Lignite was remarked by Richters to absorb oxygen without an equivalent development of carbonic acid. This is doubtless due to

* Dingler's Jour., vol. 190, p. 398.

† Bischof, Chem. Geol., i, 770.

the excess of hydrogen in lignite, and the same fact observed in the case of mineral coal is to be referred to a similar but smaller excess of hydrogen.

Richters has succeeded in establishing experimentally the principles of the oxidation of the organic parts of coal, as follows:

(1.) Its property of absorbing oxygen at a moderate temperature essentially depends on its proportion of disposable hydrogen. This, together with a certain proportion of carbon, is oxidized with the formation of water on the one hand, and carbonic acid on the other. Some unknown modification of the fixed carbon of the coal by oxidation is also apparent, as shown by increase of weight.

(2.) The carbon of coal available for conversion into carbonic acid at temperatures not above 190° C., does not commonly exceed 5 to 6 per cent. The rest of the carbon exhibits little or no attraction for oxygen within the same temperature.

Freshly won coals were observed by Richters to absorb oxygen at first with great avidity—a fact which led this chemist to distinguish between a physical and chemical absorption of oxygen. The degree of the former was found to be in relation to the widely different hygroscopic capacity of the coal, which proved to be independent of its structure. A condensation of oxygen is supposed by Richters to precede the chemical oxidation of coal, and the amount of oxygen capable of being thus physically absorbed under the same conditions to be measured by the amount of hygroscopic water which the coal is capable of absorbing after having been dried at 100° C. How this could well be other than a mere coincidence in the presence of hygroscopic water, which even freshly-mined coal can scarcely be without, does not seem clear. It is shown that carbonic acid is likewise evolved most rapidly at first in the case of freshly-mined coal, a fact which, contrary to the suggestion of Richters, seems to point to chemical rather than to physical absorption of oxygen.

The oxidation of coal was proved by Richters to be accelerated by elevation of temperature, and a moderate temperature long continued to produce the same effect as a higher temperature for a shorter period.

Assuming the physical absorption and condensation of oxygen preliminary to oxidation of the organic parts of coal, Richters attributes elevation of temperature in masses of certain coals under favorable circumstances chiefly to the heat developed by both processes of absorption. Yet by a great number of experiments with

five different coals freed from air, he found that in twelve days only one of the number absorbed more than three times its own volume of oxygen, another one $2\frac{1}{2}$ volumes, and still another only $1\frac{1}{4}$ volume.*

In order to show that the heat developed by such a degree of oxidation is sufficient to raise coal under favorable circumstances to a high temperature, even to the point of self-ignition, Richters instances the well-known tendency of certain kinds of charcoal, such as that prepared for gunpowder, to ignite spontaneously from its rapid absorption and condensation of oxygen.

The quantity of oxygen absorbed by two of the coals experimented upon by Richters proved to be about six times the coefficient of absorption of oxygen by boxwood charcoal, as determined by Saussure, at 12° C. and under a pressure of 724 mm. of mercury, viz., 9.25 (9.4.)†

Assuming that such a degree of absorption would be capable of raising bituminous coal to the temperature of ignition, which according to Violette is between 400° and 600° C., and also assuming that six times less absorption is attended with six times less heat, Richters finds that the temperature of the coal would be raised by the absorption of oxygen alone to 83° C., from which some deduction from loss of heat is necessary. If it were safe to follow this mode of reasoning any deficiency of temperature might be further assumed to be met by progressive absorption of oxygen from progressive elevation of temperature, and thus the degree of heat necessary for spontaneous ignition to be attained—under favorable, if not, indeed, ordinary circumstances.

Richters reaches a similar conclusion in another way. Having shown that at a temperature of 70° to 80° C. three different coals lost in fourteen days as high as 3.6 per cent. in calorific value, the same effect is assumed to be produced in only two or three days at 105° C., or in less time at a still higher temperature, and of course with less loss of heat attending the process of absorption of oxygen in proportion to the reduction of the time. Theoretically computing the heat of combination in the case of one of these coals, the total amount of heat evolved by a pound of coal was found to be represented by 286 calories. Had these heat units been exclusively used to heat the coal its temperature would have far exceeded the point of ignition.‡

* Dingler's Jour., vol. 195, p. 452.

† Watts, Dict. of Chem., ii, 804.

‡ Dingler's Jour., op. cit., 453.

Hence, according to Richters, coals possessing the highest power of absorption are those which likewise oxidize most rapidly and evolve the greatest amount of heat, and thus under favorable circumstances exhibit the maximum effects of weathering. Circumstances especially favorable for the production of such effects are (1) a porous condition of the coal or a state of small division, and (2) the accumulation of coal under such conditions as to prevent radiation of heat, as when stored in confined spaces or in large masses. While granting that the heat evolved from the oxidation of pyrites may in coals of low absorptive power be greater than that evolved from their absorption of oxygen, especially under favorable conditions of moisture, Richters adduces evidence to show that coals most liable to spontaneous ignition are not those which contain most pyrites. This evidence has been arranged by Percy in a tabular form,* and tends to prove this point alone, while it suggests the probability that some measure of the self-inflammability and, *pari passu*, the weathering property of coal is afforded by its degree of hydration, as this is seen to rise in the order of self-inflammability.

The causes of the evolution of a high degree of heat in coal and its evolution to a moderate extent differ only in degree. Self-ignition of coal therefore may be regarded as the climax of weathering action. In the present imperfect state of our knowledge on the subject, such extreme effects of heat are alone sufficiently marked and well observed to afford positive facts as a basis for a theory, not only for this high degree of heat itself, but also for the less elevated temperatures through which, as shown by Richters and others, all considerable effects of weathering as distinguished from the self-ignition of coal are either directly or indirectly produced. Such effects, however important, are from their nature generally but imperfectly observed, and, indeed, are capable of estimation only by exact and very laborious investigation.

Having proved the evolution of high degrees of heat in coal without the intervention of the heat of combination developed by permutations succeeding the decomposition of pyrites, Richters has endeavored to show that the *average* proportion of pyrites in coal is ordinarily too low to be capable of causing more than a slight and accessory elevation of temperature. The proportion of 1.01 per cent. of pyrites contained in Upper Silesian coal, according to Grund-

* Fuel, 299.

mann, is made the basis of a calculation of the approximate elevation of temperature through the oxidation of this proportion, and found by Dulong's rule to be 72° C., without loss of heat upon the assumption of immediate action.*

This reasoning assumes that pyrites is uniformly distributed through the coal, while it is obvious enough that this, scarcely without exception, is far from the fact. A local concentration of pyrites, or its accidental accumulation in any one spot, varies the function of any computation of this kind to such an extent as to lead to results which would necessarily imply self-ignition of coal, even under ordinary circumstances. Spontaneous ignition of coal in any case is doubtless an action at first limited to contracted portions of the mass, and may not involve uniform conditions of temperature throughout any considerable portion of it. A spark, indeed, here as usual, is a full accomplishment of the act of ignition.

The heat of combination evolved by the permutations having their origin in the decomposition of pyrites, includes not only that of its oxidation, but that also of the formation of sulphates, and of the hydration of these sulphates as well as of the hydration of the excess of sulphuric anhydride. Heat from these additional sources did not enter into the above estimation by Richters of the calorific effect of the vitriolizing of pyrites in coal.

Richters thus considered the two concurrent and co-operative processes of evolution of heat in coal, namely, the oxidation of its essential or organic constituents on the one hand, and the oxidation of its accessory pyrites on the other, as separate, and to a certain extent independent actions, which in point of fact are, together with their effects, practically inseparable. However capable may be the oxidation of the organic portions of coal of causing elevation of temperature sufficient for the production of the extreme as well as the less exceptional effects of weathering action, it can hardly be doubted that as an auxiliary source of heat the oxidation of pyrites must play an important if not an equal part. The mechanical results of pyritic oxidation in effecting disintegration and comminution of the coal, cannot fail to be recognized as among the most favorable conditions of weathering action, by increasing the perviousness of the coal and hastening its oxidation; or, in other words, by diminishing the loss of heat. Moreover, it is the physical rather than the chemical results of weathering action which are the more commonly ap-

* Dingler's Jour., op. cit., 450.

preciable as well as practically the more detrimental. The limits of the loss in calorific power, of which most coals seem to be susceptible without air-slacking, are far within the destructive effects of disintegration in the case of coals for certain purposes. If, as appears probable, physical disintegration of coal may be taken as some measure of the extent of oxidation of its essential portions, a reduction of coking power would be a practical result, even where the small division of coal might be no detriment.

The powerful action of the oxidation of iron pyrites in effecting the disintegration of rocks is well known, as well as the almost universal distribution of this substance in one or the other of its dimorphous conditions in rocks of all kinds. This action is partly chemical and partly mechanical. Through the development of sulphuric acid by the oxidation of pyrites in moist air, a corrosive or dissolving effect is produced upon neighboring particles of rocks, especially upon carbonates, phosphates and some silicates, while mechanical disintegration follows from the generation of heat from the increased porosity of the rocks, and from their consequent greater susceptibility to other agencies of weathering action, such as atmospheric oxidation, sunlight and frost. But this action is not confined to particles of rocks in immediate contact with decomposing pyrites, as the same effects are extended over a wide range by the solution of sulphuric acid, and also, as we shall see in the sequel, of ferrous sulphate in circulating waters. Such weathering action of pyrites in rocks is seen to be greatest when the pyrites is in the form of marcasite. As compared with yellow pyrites or pyrite, this form of bisulphide of iron is exceedingly unstable, especially under the combined influence of moisture as well as of air.

Bisulphide of iron is converted by the atmosphere, aided by moisture, into ferrous sulphate (green vitriol). As the original pyrites consists of two parts of sulphur and one part of iron, while but one part of sulphur enters into the composition of ferrous sulphate, the remaining part, in the oxidized state of sulphuric acid, is left free to exercise its own affinities for other mineral substances with which it may come directly or indirectly in contact. Some of the most extensive and important reactions between mineral substances within the upper part of the earth's crust have their origin in the above simple reaction. Thus the effect of sulphuric acid set free by the weathering oxidation of pyrites is to convert limestone into gypsum, dolomite into the same and magnesic sulphate (epsomite); common salt, into sodic sulphate (glauber salt); talcose slate into glauber salt

with the separation of silica; clay slate into aluminic sulphate (halotrichite) and other hydrous sulphates of alumina (alums), also with the separation of silica; etc. The heat developed by the oxidation of pyrites in addition to the heat caused by the hydration of sulphuric anhydride from the atmospheric vitriolizing of pyrites, is sometimes sufficient to cause the spontaneous ignition of coal in masses or even in its native bed. The conversion of the more bituminous class of coals into natural coke, anthracite, or graphite may be supposed to be in some cases through the generation of heat caused in the same way.

Ferrous sulphate passes readily into solution in circulating waters, and is then capable of producing reactions similar in effect to those produced by sulphuric acid, by exchanging acids with many common mineral substances with which it may be brought into contact. Thus the result of a solution of ferrous sulphate upon pure limestone is calcic sulphate (gypsum) and ferric carbonate (siderite), passing by further oxidation into hydrous ferric oxide (limonite). Upon impure limestone (argillaceous limestone) the result is gypsum, ochreous clay, and iron ochre. Upon decaying organic matter or bitumen the result is a reduction or deoxidation of ferrous sulphate to ferrous sulphide (FeS) and carbonic acid. The effect of the same solution upon crystalline rocks containing, for example, oligoclase (and generally also particles of pyrites, sometimes in a state of decomposition), is the formation of gypsum and hydrous aluminic sulphate (alum) on the one hand, and ferric carbonate and ferric oxide on the other. And so on through a great number of more or less familiar reactions, varied by circumstances of association and contact in the earth's crust.*

Similar reactions take place in mineral coal through atmospheric decomposition or weathering of pyrites, both *in situ*, and after extraction. Solutions of sulphuric acid or of ferrous sulphate, in contact with the original miscellaneous sedimentary matter of coal seams, perform within them numerous transformations of mineral substances, resulting in both soluble and insoluble salts. Of these the more soluble are often diffused, and sometimes disappear through the action of circulating waters, but they may under favorable circumstances be retained between the cleavages and laminæ of the coal, or appear as an efflorescence upon its surface. The number and variety of these compounds due to weathering action (epigenesis) become

* Senft, Synopsis d. Mineralogie, 345.

greatly extended by the circulation within the coal strata of mineral waters containing sometimes in solution comparatively rare salts, which may give rise to new combinations according to their contact with other mineral salts freely present, as we have seen, in coal seams. Some of these combinations are difficult to identify as separate mineral species, owing to the minuteness of their crystallization and the impracticability of isolating them. The following list includes some of the more common and distinctly recognized species of non-hydrogenous minerals found in coal seams, and believed to be products of either simple or complex weathering, as defined by Roth,* and more or less connected with the above briefly indicated chain of reactions:

Sulphur.	Löwigite.
Silica (Quartz, Opal, etc.).	Kaolinite.
Mellite.	Siderite.
Humboldtine.	Limonite.
Phosphorite.	Hematite.
Allophane.	Melanterite.
Halite.	Coquimbite.
Calcite.	Vivianite.
Dolomite.	Sphalerite.
Gypsum.	Galenite.
Barite.	Cuprite.
Mirabilite.	Cinnabar.
Epsomite.	Millerite.
Keramohalite (Alunogen).	Erubescite.
Halotrichite.	Erythrite.
Copiapite.	Malachite.†

Marcasite appears to be the prevailing form of iron pyrites in unaltered or sedimentary rocks, including bituminous coals and lignite, while yellow pyrites or pyrite is the prevailing mode of its occurrence in crystalline and metamorphic rocks, including anthracite. This difference affords one reason why bituminous coal yields far more readily than anthracite to the weathering action of its pyrites, although its chemical and physical constitution is also far more favorable to the oxidation of its organic constituents. Some of the anomalies observed in coals of transition types in respect to resistance to weather are perhaps as much due to differences in their accessory pyrites as to the quantity contained by them. The occurrence of such differences, however exceptional, among bituminous coals seems adequate to explain the cases on record in which excessively pyritic

* Chem. Geol., 69 et seq.

† Mietzsch, Kohlenlager, 87.

coal of this class has resisted pyritic decomposition under the same conditions known in other cases to induce, not only energetic pyritic weathering, but spontaneous ignition of the coal. Any doubt as to this point proceeds from the want of specific identification of the pyrites in different coals under such circumstances as to render it a question of importance.

To whatever degree the formation of the numerous derivative products of weathering action may be capable of physical effects upon mineral coal, it is certain that the most commonly occurring of these products exert the most powerful physical as well as chemical influence of them all, namely, those proceeding directly from the oxidation of iron pyrites, especially sulphuric acid, green and white vitriol, and the series of hydrous sulphates, including soluble double salts of alumina with other bases. Other metallic sulphides when present as rare occurrences in mineral coals under circumstances favorable to weathering, must also exert an influence only second in power to that of an equal quantity of iron pyrites.

The volumes of these oxidized products being much greater than that of the original pyrites, its decomposition is necessarily attended by disintegration of the coal. Moreover, attraction of water by sulphuric anhydride, together with the oxidation of the pyrites itself, and the hydration of the resulting salts of iron is attended by elevation of temperature which, giving rise to expansive force, promotes the tendency of coal to fall to pieces or to "slack," besides rendering it all the more pervious to atmospheric oxygen.

Heat is also found to increase the action of oxygen on all the other alterable constituents of the coal. Hence the weathering produced in coal by the oxidation of its carbon and hydrogen is accelerated and promoted by elevation of temperature, though not by moisture. On the other hand, in the weathering effect of the oxidation of pyrites chemical action is favored by moisture, and in a greater degree by moisture combined with elevation of temperature.*

Thus it will be seen *a priori*, that besides the weathering effect of its immediate oxidation, an excess of pyrites in coal tends to produce rapid oxidation of the organic as well as the inorganic parts of the coal and mechanical disintegration of the mass, especially through its propagation of heat. Heat is also developed by the oxidation of the organic constituents of the coal. Such a progression of effects serves to explain some of the more energetic phenomena of coal-

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* Percy's Fuel, p. 290 *et seq.*

weathering, such as complete air-slacking, loss of caking or coking power, development of heat and spontaneous ignition.

Coals rich in pyrites were found by Richters to yield the more readily to oxidation in a damp state. Coals poor in pyrites, on the other hand, absorb oxygen with the greatest facility in a dry state. Pyritic coals are, therefore, best preserved in a dry place, and non-pyritic coals of high absorptive powers in a damp place. Thompson, without distinguishing such differences of effect as determined by ratios of pyrites, regarded the relative action of "dry-rot" and "wet-rot," so-called by him, as 10 to 13 respectively.*

The proportions of hygroscopic water absorbed by different coals, under the same conditions, vary, according to the observations of Richters, within the wide limits of 2 to 7½ per cent., as above remarked. The power of coal to absorb gases was found to be practically the same as its capacity to absorb water.

Richters states that the absorption of oxygen by coal, not freshly won, of low absorptive power, is retarded and even suspended by the action of sunlight. Cloez, on the other hand, states that light facilitates and darkness retards weathering of coal.†

Among the ordinary circumstances favorable to the weathering of coal after winning is especially its accumulation without ventilation in stock piles of great magnitude at mines, points of shipment, or upon the premises of dealers, so as to admit of retention of the heat developed and to promote the process of oxidation under the influence of meteoric water and the heat of the sun. Coals of certain kinds or in certain conditions, exposed in this manner to the weather, are well known to develop heat, even to the point of spontaneous ignition, while other kinds of coals, especially those low in pyrites, may develop the greatest quantity of heat under protection from the weather.

Coal in its native bed may be subjected to weathering action, indefinitely continued, by lack of impervious covering, especially in isolated hills above the level of superficial drainage. Or, parts of coal seams adjoining excavations in mines may likewise be subjected to a considerable degree of weathering.

The outcropping portions of coal seams are almost invariably found to have suffered more or less deterioration, including in the case of coking coal loss of coking power.

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* Thompson's memoir appeared in the London Journal of Arts, 1865; Dingler, op. cit., 162. See tabular exhibit of the results of Richters, as arranged by Percy, Fuel, 294.

† Kerperly, Bericht, 3–4, 32.

Gob inside of mines and screenings outside are well known to weather to the point of spontaneous ignition.

With the object of determining the actual effect of the weathering of coal under ordinary circumstances, Richters introduced, by means of baskets, several hundredweight of freshly won coals, from different mines, into a large stock-pile of coal unprotected from the weather, and after drying at 100° C. After exposure varying from nine to ten months in some instances a loss of weight from 0 07 to 0.47 per cent. was observed, and in others a gain from 0.2 to 2.16 per cent. The loss of hydrogen proved, with a single exception, very low in each case, and accordingly any reduction of coking power and calorific value unimportant. A slight increase of the quantity of coke was observed and a corresponding decrease of the volatile constituents. These alterations, however, all proved too slight to affect injuriously the practical value of the coal. These practical experiments were in agreement with those of Reder, who, without appreciating the effect of oxidation in increasing the weight of the coal, was disposed to refer a similar result to errors of observation.

The coking power of coal is reduced by weathering, and, as it is well known, may even wholly disappear from certain coals originally low in disposable hydrogen, and therefore of low coking power. Whether the coke be reduced or increased by weathering is obviously determined by the proportion that the loss of hydrogen bears to the gain of oxygen. The coking power of coal rises, unlike the quantity of coke, with the proportion of hydrogen. The one indeed seems to be nearly in inverse proportion to the other. Whether the ashes of the coal be increased or reduced by weathering depends on similar relations to each other of loss of hydrogen or gain of oxygen. In the case of coking coals their proportion stands in close relation to the quantity of coke produced.

Freshly-won coals, or those namely the richer in disposable hydrogen, are commonly preferred for gas-making above weathered coals, whose proportion of combined or indisposable hydrogen is increased at the expense of the disposable hydrogen, and, as suggested by Richters, resulting in a reduction of the quantity of available gaseous matter contained in it. The illuminating quality of the gas evolved at the particular temperatures employed probably determines this preference rather than the quantity. By the use of weathered coal the water passing into the gas is apparently increased. On the other hand, the quantity of sulphur combinations, or at least the sulphurous acid is reduced.

Fleck, however, measures the adaptability of coals for gas-making by their quantity of combined hydrogen, and distinguishes as gas-coals *par excellence* those which contain 1000 parts of carbon by weight to at least 20 parts of combined hydrogen. If this view of the question be right, it is obvious that, other conditions being equal, weathered coals containing the greatest quantity of combined hydrogen would be the best for gas-making, but for the probability that a less proportion is available for gas at the ordinary temperatures employed in the process.*

The practical effect of the weathering of coal on the value of the fuel for metallurgical purposes, varies with the quality of the coal. In the case of anthracite, the weathering of which, so far as known to the writer, has not received special investigation, it appears probable that the only appreciable results within the ordinary limits of exposure of stocked coal are confined to the oxidation of the accessory pyrites, and that but to a slight degree, as the pyrites is present in the more stable form of pyrite. In the case of coking coals, however, the tendency of weathering action, while reducing and finally destroying their coking power, is to convert the accessory pyrites, usually present in the unstable form of marcasite, from the hurtful state of bisulphide into comparatively innocuous sulphates. The same is true in respect to dry, non-coking, or block coals, which, while they possess little or no coking power, and therefore suffer no practical deterioration in this respect, are nevertheless often deprived of their coherence through the disintegrating tendency of the oxidation of pyrites. The calorific value of all varieties of coal must be supposed to be in inverse proportion to the extent of weathering action. If weathering action be considered alone in its effect on pyrites the result is analogous to the weathering oxidation of pyritic iron ores, which affords one of the means employed in Europe of preparing them for the blast furnace. Sulphurous coke, likewise exposed to weathering action, necessarily improves in respect to the form of its sulphur compounds. However rich in sulphur, therefore, a weathered coal or coke may appear, it is sometimes important to distinguish between its relàtive proportion of harmful and harmless sulphur. On this subject reference may be had to a previous paper in these Transactions, also read at the Montreal meeting.†

* Steinkohlen Deutschlands, 240, 265.

† On the Relations of Sulphur in Coal and Coke, by the writer. This volume, p. 181. See also Preussische Zeitschrift für Berg. Hütten u. Salinenwesen, xix, p. 96.

However slightly anthracite may be supposed to be affected by ordinary exposure to the weather, the outcropping portions of seams of this variety of coal are frequently found to have undergone secular weathering, as seen in the partial oxidation of its pyrites, which has sometimes resulted in physical disintegration. Similar visible effects are not unfrequently to be observed in the pillars of old anthracite collieries, but are much more common in bituminous coal seams, and here rendered all the more marked from the greater influence of weathering action upon this variety of coal. Among such effects, requiring no special investigation to be perceived, are reduction or loss of coking power in the coals, disintegration, or "air-slacking" of pillars, and the coating of walls or surfaces of coal with efflorescences of salts, the result of chemical permutations chiefly through the agency of oxidizing pyrites, as above described.

Pillar coal from the mines of the Cambria Company at Johnstown, Pa., although standing for only two or three years, when recently drawn was found to retain so little of its original coking power as to require admixture with freshly-won coal for the production of satisfactory coke.* The Kemble Coal and Iron Company has recently had a similar experience with outcrop coal at R'ddlesburg, Pa., while the coke from sound coal obtained from the interior of the same seam leaves nothing to be desired. The coke made for the supply of the Fairchance furnace in Fayette County, Pa., is from an area of the Pittsburgh coal seam which is without a rocky cover. This coke is far below the standard of Connellsville coke, made from coal of the same seam under ample cover, the coal itself on the Fairchance tract having undergone a high degree of pyritic weathering, whence a corresponding oxidation of the organic part of the coal may naturally be inferred.

The chemist of the present geological survey of Pennsylvania has recently published a series of comparative proximate analyses of coals, under different dates, by way of affording an approximate estimation of the degree of weathering sustained by certain well-known coals of that State.† While not claiming further results than obviously follow from such an incomplete investigation of the subject, he has omitted to state what are the conditions of relation between the analyses of different dates. So far as the facts here exhibited by Mr. McCreath go, they are in agreement with the theory of the oxidation of coal.

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* Report MM, Geol. Surv. Pa., 120.

† Report MM, 121.

In the course of the above remarks I have distinguished the weathering effects in coal as produced (1) by the oxidation of the coal itself, and (2) by the oxidation of its accessory pyrites. The effects of the first process thus distinguished are seldom appreciable, except in the case of coking coals through a reduction of their coking power, without a close comparison of analyses of the same coal, executed immediately before and immediately after the period in which the change may have taken place. The effects of the second process, on the other hand, while capable of estimation only by a similar quantitative comparison, are generally attended with such physical changes as at least to become appreciable. While the first process may proceed without the co-operation of the second to any marked degree, the second, under ordinary circumstances, can scarcely become appreciable in its effects without having to a greater or less degree involved the first. This is in consequence of the elevation of temperature caused by the oxidation of pyrites, and by the heat of combination resulting from further permutations between the active constituents of this accessory substance and the miscellaneous or other accessory matter with which these may come in contact, besides oxygen and water, whose powerful influence continues as long as any pyrites remains undecomposed.

It may readily be supposed, however, that the oxidation of accessory pyrites is sometimes greatly promoted by the very circumstances which result in the greatest loss of heat, and hence that oxidation of the coal itself under such circumstances may not always be a necessary consequence of pyritic weathering. When, on the other hand, pyritic decomposition takes place under more favorable circumstances for the retention of heat, as when coal is stocked in large masses, or as in the case of the accumulation of gob in mines or of culm upon the surface, the oxidation of the organic parts of the coal must necessarily be involved.

Numerous instances of coal *in situ* vitiated from exposure to superficial agencies of secular weathering are to be recognized in all coal-fields, especially in such as present the coal strata in steep dips so as to bring edges of coal seams to the surface; or, again, in areas of coal strata where these are warped into folds and bosses so as to bring the planes of portions of coal seams near the surface above local drainage or water level. In the latter case both the stratigraphical and topographical features conspire to produce conditions favorable to a high degree of weathering.

General denudation and local erosion often determine the topo-

graphical configuration of the surface in such cases, while the special stratigraphical features are a part of the general geological structure. In the Bristol coal-field, at the base of Mendip Hills, the coal is weathered to a depth of 80 metres to such an extent as to be practically worthless. In the highly disturbed coal measures of Central France portions of coal seams have equally suffered from similar causes.*

One of the most remarkable instances of the weathering of coal *in situ* under the conjoint influence of stratigraphical and topographical configuration, is exhibited by the Coalton or No. 7 seam, so-called, of the Kentucky series of coals at Willard, in the Hanging Rock district of Eastern Kentucky. The coal of this seam is locally a dry, non-coking and tenacious variety, well known in the region and highly esteemed as a blast-furnace fuel in the raw state, being used as such in both the Kentucky and Ohio divisions of this district. On the tract of the Belfont Iron Company, at Willard, it was opened in 1873 in an isolated hill, where it occurs above water-level, covered only by a body of shaly strata. This covering exhibits the usual weathering of superficial strata in this latitude, unprotected as in higher latitudes by drift. Under such circumstances the coal seam in this hill exhibits, by means of the workings which extend through it, a high degree of pyritic weathering, attended by disintegration, which may be taken as an index of the further oxidation or weathering of the organic portion of the coal. The coal proved unsuitable as a fuel for the blast furnace and the workings have been discontinued. As the seam in this hill proves thinner than in all the surrounding outcrops where it occurs under good cover, a shrinkage or contraction has here probably taken place throughout the plane of the seam similar to what is often elsewhere observed in exposed outcrops or edges of coal seams.†

The theory of the oxidation of coal advanced by Fleck and elaborated by Richters, is adequate to explain all the phenomena of weathering hitherto described by different observers. In the course of these remarks I have endeavored to show that a greater importance than is admitted by Richters is to be ascribed to the oxidation of the accessory pyrites, as a part of the complex phenomena of coal-weathering as ordinarily observed, and also to the sequence of chemical permutations in coal which the oxidation of this accessory substance involves under favorable circumstances. The means for the estima-

* Mietzsch. Geologie der Kohlenlager, 222.

† See appendix to this paper.

tion of the action of decomposing pyrites both as a cause and effect in the process were necessarily excluded from the conditions of the experiments performed by Richters. And it may be questioned whether the extent of its participation in the process can be determined by experiments on a small scale, or otherwise ascertained than by observation upon large masses of coal under the incessant vicissitudes of exposure to the weather. Nor is the full extent of its alterative power capable of estimation at all when exerted for indefinite periods, as in the secular weathering of coal seams. It can scarcely be doubted, it seems to me, that the oxidation of the accessory pyrites, especially in its least stable form of marcasite, is a common cause of the elevation of temperature in coal, even within short periods of time, and that as such it is no less powerful, at least, than the physical and chemical absorption of oxygen by the coal itself.

The whole series of phenomena blend on the one hand with the occlusion of certain gases in fresh coals, and on the other hand with the varying conditions of spontaneous ignition of coal. These conditions are outside of the present subject. I have not undertaken to discuss them, except so far as the self-ignition of coal is to be regarded as an exhibition of extreme elevation of temperature, on which the oxidation of coal mainly depends, and therefore as a phenomonon incidental to its weathering, and as the climax of this process.

APPENDIX.

With the object in view of obtaining data whereby the extent of the weathering sustained by the "Coalton" or "No. 7" seam of coal, at Willard, Ky., under the circumstances briefly described above, might be approximately and comparatively estimated, Dr. Robert Peter, Chemist to the Geological Survey of Kentucky, kindly undertook, at my request, the following interesting investigation.

Mr. George Gibbs, of Riverton, who is very familiar with the topographical geology of the Hanging Rock district of Kentucky, and under whose superintendence the No. 1 Entry at Willard was opened in 1873, selected on the 6th of November, 1879, samples of coal intended to illustrate:

1. An average of the coal without the top bench. Sample A.
2. The most highly weathered coal to be found inside the mine—not from the top bench (which as a pyritic coal it is a custom of the region to reject). Samples B and C.

3. The least weathered, or brightest and soundest coal from the bottom bench. Sample D.

The following is Dr. Peter's description of the different samples:

1. A. A handsome, pitch-black, shining, pure-looking coal, iridescent on some of its surfaces. Some pieces show imperfect lamination, with very little fibrous or granular coal between the irregular laminæ. Other portions break into irregular fragments with highly polished surfaces. A *few* minute crystals and irregular scales of pyrites are visible in the fibrous or granular coal (between partings), but no appearance of saline efflorescence. From No. 1 Entry, mouth of left cross entry.

2. B. Outcrop coal. Highly weathered. Friable, showing saline and ferruginous efflorescence on the surface of lumps. Some visible pyrites. From two lower benches, No. 1 Entry, twenty feet from west end.

C. Outcrop coal. Same as B, with the exception of the absence of visible pyrites. From Buzzard's Roost Entry, at Willard, one-half mile northeast of No. 1 Entry, where the coal is under good cover; twenty-five feet from mouth of entry.

3. D. A bright, pitch-black coal, iridescent on the surfaces, showing very imperfect lamination (sub-conchoidal), breaking with lustrous surfaces. Mineral charcoal or fibrous coal on one of the partings. Scales of pyrites on some of the surfaces. From bottom bench, No. 1 Entry, mouth of left cross entry.

4. For comparison. E. A jet-black, pure-looking, iridescent coal, with but little fibrous coal or pyrites. Including top (pyritic) bench. Averaged by P. N. Moore. No. 1 Entry. (*Report Geological Survey of Kentucky*, New Series, Vol. I, pp. 183 and 184, 1876.) Known as "Coalton Coal," or "No. 7."

F. Same as above, but dull black.

The low proportion of hygroscopic moisture in E and F, or in the specimens collected by Mr. Moore, as compared with that in Mr. Gibbs's samples, is attributed by Dr. Peter to the longer time during which the former were exposed to the dry atmosphere of the laboratory. C, however, contains an inordinate proportion, due to excessive weathering.

The greater proportion of ash in Mr. Moore's samples, E and F, is doubtless to be referred to contamination by the upper objectionable bench, usually rejected in mining.

The excessive proportion of sulphur in the same samples is also to be referred to the same circumstance.

	Average, exclusive of top bench.	Weathered coal, friable.		Firm and sound.	Average from run of mine, including top (pyritic) bench.	
	A.	B.	C.	D.	E.	F.
Specific gravity	1 320	1.347	1.349	1.302	1.352	1.337
Hygroscopic moisture	6.20	6.10	8.06	6 80	3.20	3.70
Volatile combustible matter	34.20	32.80	33.30	36.90	35.06	35 96
Coke	59.60	61.10	58.64	56.30	61.74	60.34
	100.00	100.00	100.00	100.00	100.00	100.00
Total volatile matter	40.40	38 90	41.36	43.70	38.26	39.66
Fixed carbon in the coke	55.40	54.30	52.52	52.44	54.40	52.94
Ash	4.20	6.80	6.12	3.86	7.34	7.40
	100 00	100.00	100.00	100.00	100.00	100.00
Percentage of sulphur in the coal	2.104	2.172	1.167	1.965	2.631	2 727
Percentage of sulphur in the coke	1.192	1 219	.854	1 261	Not determined.	Not determined.
Percentage of sulphur lost in coking	0.912	.953	.313	.704	" "	" "
Percentage of sulphur soluble in bisulphide of carbon (free sulphur)	0.060	.030	.015	.038	" "	" "
A′. Sulphuric anhydride, soluble in water	0.198	0 266	0.470	0.139		
B′. Sulphuric anhydride, soluble in water, slightly acidulated	0.059	0.092	0 034	0.058		
Total sulphuric anhydride	0.257	0.358	0.504	0.197		
Iron sulphates in A′	0.243	0.465	0.026	0.227		
Calcic sulphate in A′ and B′	0.039	0.032	0.665	0 032		
Magnesium sulphate in A′	0 015	0.013	0.099	0 009		
Aluminic sulphate	trace.	trace.	trace.	trace.		
Sulphur equivalent to total sulphuric anhydride	0.103	0.143	0.201	0.079		

NOTE.—For other analyses of "Coalton," or "No. 7," coal of the Hanging Rock district of Eastern Kentucky, under better conditions of preservation, see Geological Survey of Kentucky, vol. i, p. 159 (new series).

SILVER ISLET.

BY THOMAS MACFARLANE, ACTONVALE, QUEBEC, CANADA.

I. INTRODUCTION.

AMONG the industrial enterprises which have, from time to time, been undertaken in our Dominion, few have been more uniformly unsuccessful than those which have had for their object the development of our mineral resources. These resources have not been regarded as insignificant. On the contrary, it is supposed that we possess in our ore deposits something more than mere points of mineralogical interest. Some people are enthusiastic enough to believe that our mineral wealth will yet occasion the establishment in our backwoods of happy and industrious communities, causing, in fact, our wilderness to blossom as the rose. These enthusiasts do not shut their eyes to the serious obstacles which at every stage interfere with the carrying out of mining enterprises in this country, and which tempt one to believe that no such enterprise can ever be conducted to a desirable end, but they entertain the conviction that as our capitalists, miners, and technologists reach the years of discretion, and handle our mines in an economical, vigorous, and judicious manner, and as our statesmen become alive to the necessity of fostering, without extremely protecting, our metallurgical industries, mines and furnaces will gradually prosper, and so develop as to become a tower of strength to our agricultural, manufacturing, and mercantile interests.

The fact that our past experience does not generally justify such glowing anticipations makes one all the more anxious to put on record the history of a successful mine, such as the Silver Islet has, on the whole, been. My connection with its discovery and early development was a very intimate one, and I propose in this paper to sketch its history and character, and sum up the amount of silver it has so far yielded. An additional reason for the appearance of this paper is that the Transactions of the Institute contain only a few references to the Silver Islet Mine. Although this mine is reported successful, we have, of course, no guarantee that it will not fall a victim to the same influences which have

ruined other promising Canadian mines in days gone by. It is unnecessary to specify those influences. They are well known to all those who take any interest in the mining of the present day, and they are essentially the same as those which discouraged the miners of a century ago. Above the entrance of the silver smelting house, of Kongsberg, in Norway, there is to be found an inscription upon an old stone, which had been taken from a similar position in the building which had been used for a century before the present one. This inscription dates from the time when German miners were brought north to work the Kongsberg silver mines. It is as follows:

> "Eigennutz und Müssiggang
> Ist des Bergwerks Untergang,"

which may be rendered thus:

> "Selfishness and laziness
> Spoil the mining business."

Long before 1868, the year of the discovery of Silver Islet vein, silver had been found on the northwest shores of Lake Superior. When, in 1846, the lands of the Montreal Mining Company were located, silver was reported as having been found on several of its properties. Later, the British American Mining Company worked the silver vein of Prince's Location, with the results recorded in the publications of the Geological Survey of Canada.* The revival, ten years ago, of silver mining on Lake Superior was, however, occasioned by the success of the Messrs. McKellar, of Fort William, who, from 1863 to 1867, employed themselves in exploring the neighborhood of Thunder Bay, and discovered silver at several points. One of these discoveries, at Current River, was worked by the Thunder Bay Silver Mining Company in 1868 and the year following, but with very discouraging results. It was, in all likelihood, the McKellar discoveries, together with the imposition by the Ontario Government of a tax of two cents per acre on Lake Superior mining lands, which prompted the Montreal Mining Company to begin a systematic exploration of their northwestern locations. For twenty-two years these had been allowed to lie almost entirely neglected. Several of them were indeed visited and explored by Mr. Pilgrim, of Sault St.

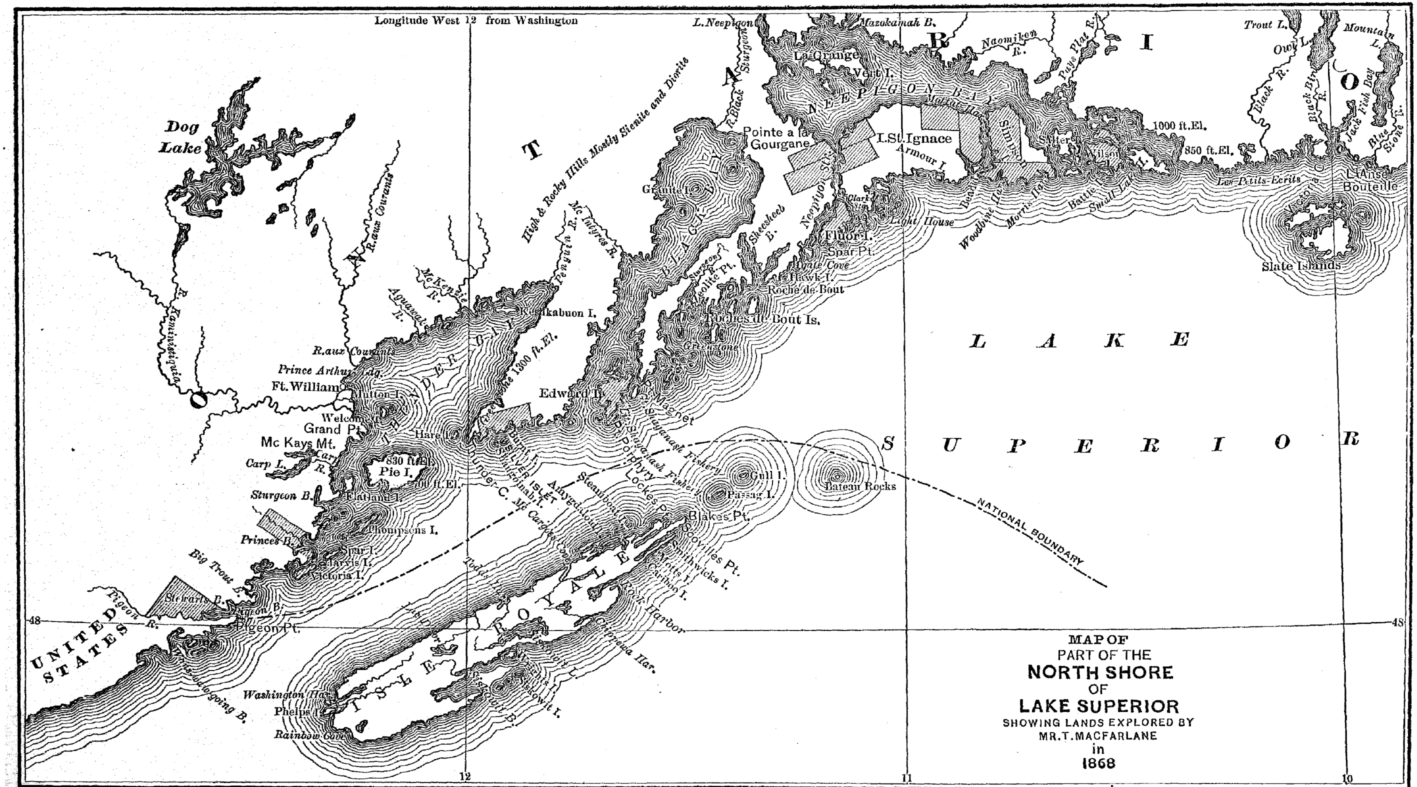
* Geology of Canada, 1863, p. 707.

Marie, and the late Mr. Herrick, P.L.S., but the results were not such as to encourage the company to proceed to active mining operations. Indeed, during the greater part of this time the company's resources were taxed to the utmost in developing and working the Bruce Copper Mines. It may safely be asserted that in doing this they experienced a dead loss of $400,000, a fact which is abundantly sufficient to account for the unwillingness of the board and shareholders to risk further capital in mining operations. The causes above given were, however, enough to induce them to incur a moderate outlay for exploring their lands, and early in 1868 I was employed by the company to take charge of a party for this purpose. During the first year I was able to visit all the northwestern locations, and it may not be uninteresting here to give a narrative of our first exploring voyage, from which have sprung such important results for Northwestern Ontario, reciting our experiences regarding the other locations, as well as the one to which Silver Islet belongs.

II. Exploration of 1868 and Discovery of Silver Islet.

On the 16th of May our exploring party, consisting of six men besides myself, arrived in Thunder Bay, on board the steamer Algoma, which was heavily freighted with men and materials for working the Thunder Bay Company's Mine. After visiting the latter and the Shuniah (now the Duncan) Mine, and calling the attention of the men of our party to the appearance and characters of the native silver and silver glance produced by them, we started in our Mackinaw boat on the 19th southwestward for Jarvis's Location.* My first impressions as to the mineral resources of Thunder Bay district were not at all encouraging. With a country rock mainly composed of grayish flags and red and white sandstones, and these lying in an almost horizontal position, the chances for finding anything of value seemed very slender to one accustomed to highly inclined and crystalline rocks. I remembered, however, that the conglomerate beds of Keweenaw Point, now the most productive and remunerative for copper, had originally been undervalued by geologists, who had never before observed valuable minerals to occur in rocks of that nature, and resolved to beware of allowing preconceived ideas to interfere with the thoroughness of our search.

* With regard to this and other localities, their positions will be seen on consulting the accompanying small map of part of the north shore of Lake Superior.


MAP OF
PART OF THE
NORTH SHORE
OF
LAKE SUPERIOR
SHOWING LANDS EXPLORED BY
MR. T. MACFARLANE
in
1868
Longitude West 12 from Washington
LAKE SUPERIOR
ONTARIO
UNITED STATES
NATIONAL BOUNDARY
ISLE ROYALE
NEEPIGON BAY
High & Rocky Hills Mostly Sienite and Diorite
Dog Lake
R. Kaministiquia
R. aux Courants
Ft. William
Prince Arthur's Ldg.
Grand Pt.
Mc Kays Mt.
Carp L.
Sturgeon B.
Princes B.
Big Trout B.
Pigeon R.
Pigeon Pt.
Pie I.
830 ft.El.
Thompsons I.
Victoria I.
Edward I.
Magnet
Porphyry
Lockes Pt.
Blakes Pt.
Gull I.
Passag I.
Bateau Rocks
Roches de Bout Is.
Roche de Bout
Fluor I.
Spar Pt.
Light House
Granite I.
Pointe a la Gourgane
I. St. Ignace
Armour I.
La Grange
Vert I.
L. Neepigon
Mazokamah B.
Naomikon R.
Pays Plat R.
1000 ft.El.
850 ft.El.
Les Petits-Ecrits
L'Anse Bouteille
Slate Islands
Trout L.
Owl L.
Black R.
Mountain L.
Jack Fish Bay
Blue Stone
Rock Harbor
Chippewa Har.
Washington Har.
Phelps I.
Rainbow Cove
Washawgoing B.
Kabuon I.
Mc Intyres R.
McKenzie R.
Aguawah R.
Sheesheeb B.
48
12
11
10

From the 20th to the 31st of May, inclusive, we remained on Jarvis's Location, examining its rocks, surveying the river from its mouth to where it leaves the location, and exploring Jarvis's Island, with the numerous veins occurring there. The geology of this location much resembles that of the western part of Wood's Location,* having the same gray argillaceous sandstones and shales, intersected by dikes of different sorts. The latter have here the same general strike as on Wood's Location, and it appears evident that they run through the range of islands which extends from Shangoinah Island and Silver Islet, on Wood's Location, past the outside of Pie Island to Prince Albert or Thompson's Island; then, still further southwest through Spar Island, Jarvis's Island and Victoria Island, joining the mainland before Pigeon River is reached. On Jarvis's Island five different veins were found, and in one of them native silver and silver glance were discovered (the former by Mr., now Dr., C. O. Brown, and the latter by Mr. Patrick Hogan), specimens of which were forwarded to Montreal. The quantity appeared at the time insufficient to merit much attention, but, from the experience afterwards acquired on Wood's Location, I was induced to believe that some work upon this vein might possibly develop a larger quantity of the metal. Accordingly, in 1869, a shaft was sunk on this vein to the depth of twelve feet, in accomplishing which work the following ore was produced.

79 lbs first quality ore, containing 3.45 per cent. silver = 39.7 ounces, at $1.25,	$49.62
2483 lbs. second quality ore, containing 0.15 per cent. silver = 54.18 ounces, at $1.25,	67.72
	$117.34

On the 1st of June we left Jarvis's for Stewart's Location, at Pigeon River, where we remained until the 21st, making a very close exploration for a distance of three miles inland. The number of dikes and veins here visible induced me to anticipate the best results, but although a good deal of time was spent on some of the veins none of them yielded any valuable minerals. High rocky ranges intersect this location generally in the direction of the dikes, and between them lie valleys containing a large area of good soil, much of which will no doubt be cultivated so soon as mining operations are carried on successfully in the neighborhood.

On the 21st of June we returned to Fort William, and on the 23d

* Canadian Naturalist, IV, N. S., p. 37.

reached Thunder Cape and Wood's Location, where we remained until 31st July. From perusing the *Geology of Canada* I had, before arriving on the location, come to the conclusion that it was likely to present many interesting geological features. Here it was to be expected that the junction of Sir W. E. Logan's upper and lower groups of the Upper Copper-bearing rocks would occur, and that the many intersecting dikes and the trap overflow of Thunder Cape would be found to present interesting relations to the sedimentary rocks. I therefore determined to make a complete geological map of Wood's Location, and arranged with my assistant, Mr. Gerald C. Brown, to have the shore line accurately surveyed. It was while engaged planting his pickets on the many islands fronting the location that Mr. Brown first landed on the rock shortly afterwards named by me "Silver Islet," and observed the vein and the galena occurring in it. I then visited the island to obtain specimens of the galena and the inclosing rock, and three men were set to work to blast out some of the galena. It was while engaged working on the Islet that one of these men, Mr. John Morgan, found the first nuggets of metallic silver, close to the water's edge. A single blast was sufficient to detach all the vein rock carrying ore above the surface of the water, but further out large black patches could be observed in the vein under water, some of them with a greenish tinge. On detaching and fishing up pieces of these they were found to consist of galena, with which were intermixed spots of an oxidized black mineral, here and there tinged with green. This black substance I succeeded in reducing on charcoal, before the blowpipe, with a little borax, to metallic silver, thus exposing at once the extraordinary richness of the black portions of the vein. I deem it worth while thus to record, more particularly than heretofore, the circumstances of the discovery, on account of the celebrity which Silver Islet has since attained. The silver was discovered on the 10th of July, and on the 15th three packages of the best specimens were shipped from Fort William to Montreal, and a telegram sent at the same time to the company's secretary giving notice of the important discovery.

A marked difference is observable between the geological structure of the locations to the southwest and those to the northeast or east of Wood's Location. The conglomerate and red sandstones which constitute the eastern part of the latter are, further to the eastward, overlaid by numerous beds of different species of basic crystalline rock, most of them bearing some resemblance to those of the dikes

occurring on Wood's Location. Derbyshire's, Merritt's, Ewart's, Ferrier's, Harrison's, Turner's, and Wilson's locations all show the sandstones with superincumbent overflows of basic rock, while Hopkirk's, Lyman's, Bagg's, and McGill's are exclusively made up of the latter, together with some beds of a trachytic nature.

The first of the eastern locations, Derbyshire's, on Point Porphyry, was reached on the 31st of July. The most of the veins indicated by previous explorers were carefully examined. Native copper and copper glance were the minerals of importance observed, but only in minute specks. It is to be regretted that the mention of the occurrence of native silver on this location, contained in former reports, leaves the precise spot unindicated. We left this location on the 4th of August, but were unable to reach Fluor Island until the evening of the 5th.

From that date till the 15th of August our party camped on Hopkirk's Location, Fluor Island. Its rocks are mostly hard and compact traps, and amygdaloidal beds are rare. No economic minerals were observed in the veins or rocks of this location, excepting a few specks of native copper and iron pyrites. An excursion was made from Fluor Island to the locations on Nipigon Straits. On Lyman's Location only one vein of respectable size was found, upon which several openings had been made. A little copper glance only was observed, but the character of the veinstone somewhat resembles that of Silver Islet. On Merritt's Location no veins were observed along the shore, but at its northwest corner an old drift was found choked up with rubbish. At numerous points along the shore of Ewart's Location metallic copper was found in trappean rocks, but mostly unconnected with any regular vein or bed. At only one point was it observed that the copper ran in a streak conformable with the stratification of the adjoining rocks, and here it was in deficient quantity. Observations similar to these were made at the workings north of Point a la Gourgaune, and on the shore of Bagg's Location.

Shortly after returning from Nipigon Straits, we left Fluor Island, and on the 19th of August arrived at Harrison's Location on the east end of Isle St. Ignace. Of all the northwestern locations it is the one of which the best accounts are given in the old reports. We remained upon it ten days, part of the time at Harrison's Landing, on the south part, and the remainder at Moffatt's Harbor, on the northeast shore of the location. A day was spent in examining the shore of Simpson's Island on the opposite side of St. Ignace Straits. More excavation has been done, buildings erected, and money spent,

on Harrison's Location than on any of the others. It is, however, very difficult to find traces of any minerals sufficiently valuable to justify the expenditures which have been made here. Several shafts have been sunk eastward from Harrison's Landing, but a careful search in the veinstone from these failed to show more than a few minute grains of copper pyrites. Along the shore, in the neighborhood of these shafts, loose pieces of veinstone were found, containing copper glance, sometimes in pretty solid "prills," the source of which was found to be a vein about thirty feet distant from the shore, under water. Further search was made by us and considerable excavation done on its apparent strike to the eastward, but without success. No silver was visible along with the copper glance in the boulders, or in the vein under water. The so-called silver veins occurring to the north of Moffatt's Harbor were found to be of a peculiar description. Instead of being several feet in width, as described in the old reports, none of them exceed two or three inches. Some of them occur within three or four feet of each other, and the intervening rock has evidently been regarded as part of the vein and included in its alleged width. The three shafts sunk here were full of water and rubbish, and we had not the necessary appliances for emptying them. A thorough examination was made of the veins on the surface, and blasting done on them, which exposed finely disseminated particles of copper glance. Among the debris on the shore a small piece of veinstone was found containing native silver.

On the 29th of August we left Moffatt's Harbor, and sailing past the north ends of Simpson's and Salter's Islands, reached "North Harbor," on Wilson's Location. Here we remained fifteen days, making a close examination of Wilson's Island and of those lying to the north of it. Numerous veins were found, some of them of good width, containing calcspar, quartz, and laumontite, but no metallic minerals. In several of the amygdaloid beds metallic copper was found; once on the northwest shore of Wilson's Island it occurred in tolerable quantity, but still not sufficient to be remunerative.

We left Wilson's Location on the 14th of September, and the same day reached Morin Harbor, on Simpson's Island. Six days were employed in examining McGill's Location. Its rocks are principally hard agatic traps, with but little indication of copper, except an occasional speck in an agate geode. Numerous veins are visible, sometimes with copper glance in very small quantity. About halfway along the front of the location from Morin Harbor a good many

pieces of prehnite containing pretty solid native copper were found on the shore. These are derived from a bed of compact trap, containing large geodes with quartz, laumontite, calcspar, native copper, etc. The number of the geodes is, however, too small to render the mass of the rock workable. On the southwest end of the location, near Woodbine Harbor, characteristic basalt was, for the first time, observed, showing both straight and bent columns.

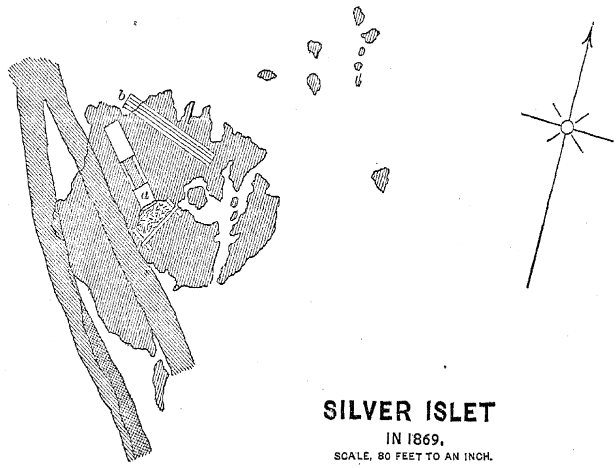
Although we were unsuccessful in discovering any valuable deposits on the territory east of Silver Islet, it would be quite erroneous to infer their entire absence. The time devoted to the exploration of this district was too short to allow of exhaustive work. Indeed, this exploration was felt at the time to be merely a reconnoissance, preparatory to more effective work after the locations had been regularly defined by a provincial land surveyor. These surveys were afterwards completed, but the explorations were not resumed. It must further be remembered, that the mineral lands in question are densely wooded, and covered by a thick layer of moss almost to the water's edge. These obstacles (which are seldom met with by Western explorers) interfere materially with a thorough search. They are sometimes removed by forest fires, and it is probable that, after the occurrence of these, a diligent "prospecting" for minerals would be well rewarded.

On account of very stormy weather we did not reach the steamboat landing, Isle St. Ignace, until the 23d of September, three days after leaving McGill's Location. The Algoma was due on the 24th, but did not arrive till the 26th. After going west to Fort William, and leaving our boat, portable forge, and other heavy articles there, we left Thunder Bay on the 27th of September, *en route* for Montreal.

III. Geology of Silver Islet and its Vein.

My plan as to a geological survey of Wood's Location was, during the following season (1869), carried to completion, and the results published in the *Canadian Naturalist* (vol. iv), accompanied by a map of the location and plan of Silver Islet, the latter of which is republished herewith. Silver Islet is situated about three-quarters of a mile from the mainland, and is much exposed to storms from the west, southwest, and east. Shangoinah Island protects it, but inefficiently, on the southwest. The island measured originally about ninety feet each way, rising about eight feet at its highest part above

the level of the lake. The whole of the rock is now inclosed and covered by the works erected for protecting and working the mine, and by new land since made. The course of the vein traversing the Islet is about N. 35° W., and its dip about 85° S.E. As



SILVER ISLET

IN 1869.

SCALE, 80 FEET TO AN INCH.

The rich part of the vein is indicated thus
a, Shaft & Shaft House; b, Slide for boat.

shown in the plan, its greatest width is on the northwest side of the islet, and it will be seen that in going southward it divides into two branches, one of which crosses the islet, and the other keeps on the west side, under water. The south part of the latter branch carried the richest ore, the eastern branch being less rich, and the whole of the vein to the northward being almost entirely barren, and consisting of a huge mass of calcspar, with quartz and occasional cubes of galena, which carry only a minute quantity of silver. Particles of silver ore were also found in some of the small "feeders" which intersect the country rock (or perhaps "horse") lying between the two veins. Fragments and masses of this rock are very often inclosed in the veinstone, and graphite very frequently associates itself with them.

The metallic minerals of the veins are silver, silver glance, tetrahedrite, domeykite, galena, blende, iron and copper pyrites, cobalt bloom, and nickel green. The two latter substances seemed to be oxidation-products of a peculiar mineral, which, in a paper published

in the *Canadian Naturalist*, and dated 1st February, 1870, I described as follows: "Besides the small nuggets and grains of pure metallic silver, there are also found in the rich ore, thin plates and grains of a sectile mineral, having a reddish-brown color, like that of niccolite, and containing arsenic, cobalt, nickel, and silver, with the latter in greatest quantity. This would appear to be a new mineral and one worthy of more minute examination." When active work was begun by the Silver Islet Mining Company, it was found that the great bulk of the silver extracted was contained in a granular mixture of the reddish-colored grains above mentioned with other minerals. The late Major A. H. Sibley—then president of the company—gave this mixture the name of "*Macfarlanite*," which is still used for it by superintendent and workmen at the mine up to the present time. In the *Engineering and Mining Journal* of 29th March last, Mr. W. M. Courtis published a paper on Lake Superior silver ores, in which this supposed new mineral is also referred to as "*Macfarlanite*." For these reasons it would seem necessary to refer to this substance more particularly. In December, 1870, I made an examination of the brown metallic grains which the rich ore above referred to leaves when pulverized in a diamond mortar, and all brittle minerals, such as calcspar, galena, etc., sifted or washed off. A blowpipe assay showed these to contain:

Silver,	78.34
Nickel,	5.98
Cobalt,	2.75
Arsenic, etc. (difference),	12.93
	100.00

I made some further trials, but was unsuccessful in separating any definite mineral.

In December last Dr. T. Sterry Hunt informed me of the discovery of the new minerals in Silver Islet ore by Dr. Wurtz, and showed me a specimen of "*Huntelite*," which seemed to me to occur in larger pieces, and to be different from the reddish grains occurring in the ore known as "*Macfarlanite*."

On subjecting the rich granular ore of Silver Islet to closer examination, assisted by Mr. W. M. Courtis, I found it to consist of the reddish-brown metallic grains, a dark-colored undetermined mineral, niccolite, galena, calcspar and quartz. Native silver in perfectly white grains or filaments is not distinctly seen. Pieces of the ore, upon being ground and polished, show the

metallic grains with a color and lustre closely resembling burnished nickel. When the calcspar, which is the principal gangue, is removed by dilute hydrochloric acid, the result is porous and coherent. The metallic minerals adhere to each other and to the metallic grains, the latter seeming to be in places coated or incrusted with the dark-colored mineral and the niccolite. Under the hammer, the metallic grains flatten out, and the glass shows that both brown and black brittle grains have separated. Still, in this process, all the metallic minerals do not seem removable, but adhere, more or less firmly, to the metallic grains.

When the ore is pulverized, and the metallic grains are freed from all brittle materials, the latter being sifted and washed off, they assay from 75 to 84 per cent. of silver. When these are further triturated in an iron mortar six different times, and the brittle matter removed by sifting on a sieve with 50 meshes to the lineal inch, the siftings thus produced assay as follows:

1st time, ...	46.41	per cent	silver.
2d "	51.55	"	"
3d "	59.25	"	"
4th "	66.24	"	"
5th "	76.1	"	"
6th "	83.7	"	"

The grains remaining upon the sieve from the last trituration have a dark-gray color and assay 92.04 per cent. silver. When roasted in a muffle they become yellowish-brown. Hydrochloric acid removes nickel oxide, and then they have the appearance of pure silver grains. In this process they lose 6.94 per cent. of their weight, and assay 95.76 per cent. silver. Triturated a second time in this manner they lose 2.12 per cent. additional, and assay 97.42 per cent. silver. From this it is evident that they assume the appearance of metallic silver on the surface only, and are not pure throughout. The grains of 92.04 per cent. when treated alone on charcoal, before the blowpipe, merely cake together, yielding a slight coating of arsenious acid. When a small quantity of borax is added, a silver button is produced, with some speiss attached, and a slag slightly tinged with cobalt oxide. Most of the nickel is removed in the speiss, but the silver still retains some of it, and upon cooling shows a greenish-gray film. This is removed by a further slight scorification with borax, and 91.33 per cent. silver is obtained. The grains of 92.04 per cent. silver dissolve readily in dilute nitric acid, yielding the following results:

Insoluble (assays 17.46 per cent Silver),	2.37
Nickel,	1.58
Antimony,	.36
Silver,	93.54
Arsenic,	2.15
	100.00

When the metallic grains first above alluded to are treated with dilute nitric acid (half acid, half water), and its action interrupted when about half the quantity is dissolved, a considerable quantity of a black powder is detached; much nickel is dissolved, and the remaining grains have the appearance of pure silver, but still showing black specks, especially in the cavities. These grains were found to consist of:

Insoluble,	1.85
Silver,	94 77
Antimony,	.85
Nickel,	.95
Arsenic (difference),	1.58
	100.00

A quantity of the metallic grains were acted on by three successive portions of very dilute nitric acid. The resulting solutions contained as follows:

1. Silver,	37.646	
Mercury,	.649	
Arsenic,	6.398	
Antimony,	.166	
Nickel,	4.661	
		49.520
2. Silver,	33.692	
Mercury,	.099	
Antimony,	.059	
Nickel,	1.220	
		35.070
3. Silver,	5.403	
Antimony,	trace	
Nickel,	trace	
		5.403
4. Insoluble quartz grains, etc.,	6.203	
Black mineral washed off,	3.756	
		9.959
		99.952

The black substance washed off from the larger grains of quartz contains antimony, lead, cobalt, nickel, and sulphur, besides 24.79 per cent. of silver.

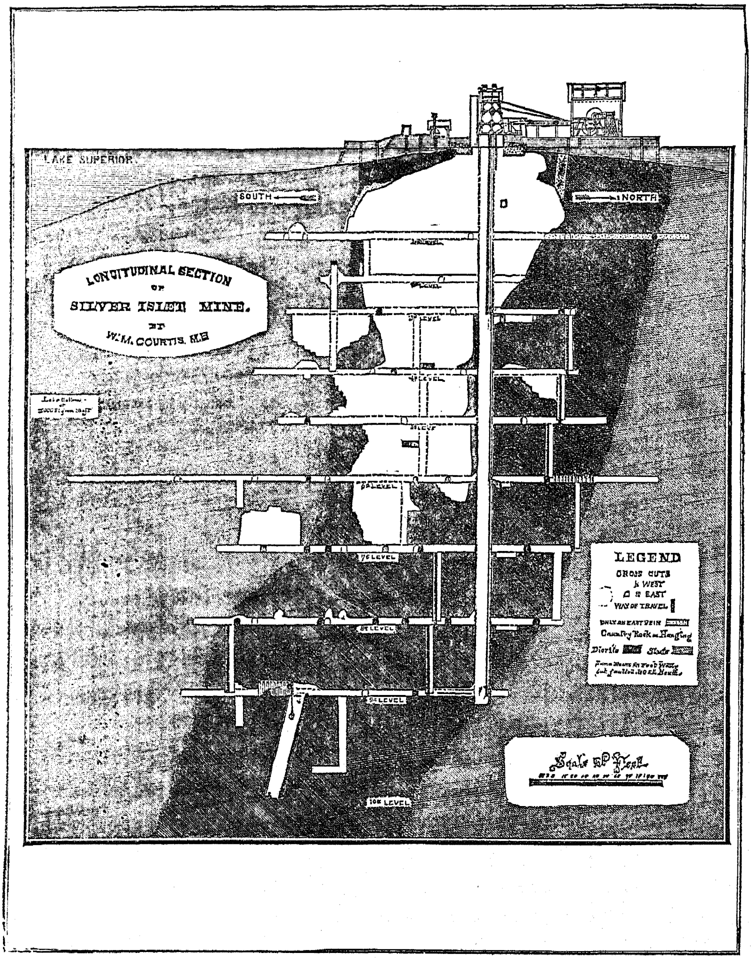
From the above experiments the great difficulty of separating the various minerals which occur in the rich ore of Silver Islet will be apparent. It seems to me to contain a great deal of its silver in the native state, which passes into the *animikite* of Professor Wurtz. Further investigation may possibly give more decided results.

With regard to the general nature of the ore yielded by the vein after sorting, one parcel of second quality—weighing 15,914 pounds, and assaying 964.2 ounces silver—was found to contain 71.5 per cent. of earthy carbonates and 14.15 per cent. of matter insoluble in acids. The relative quantity of calcareous and silicious matter varies, however, in different parts of the vein; and, in some places, streaks of quartz have preponderated to such an extent as to make some of the ore highly silicious, and much more difficult to smelt.

The rock on the Islet intersected by the silver vein is a chloritic diorite, evidently forming a dike. It differs somewhat from the rocks of the other dikes of this location, among which may be mentioned corsyte and anorthite porphyry. Judging from their manner of occurrence, it did not appear likely, when the discovery was first made, that the diorite of Silver Islet would be found to have a greater breadth than two hundred feet on the length of the vein. Outside of this distance it was anticipated that the vein would be found to intersect the gray flags, which fill out all the spaces between the various dikes. It was further thought unlikely that the Silver Islet vein, on reaching these flags, would exhibit the same degree of richness as previously. The continuation of Silver Islet vein across Burnt Island (which is identical with one shown on an old map of the location by Mr. Wilkinson), and also further inland, was traced out in 1869. It has been exposed at several points where it crosses the sedimentary beds, but there it is split up into numerous thin veins of quartz, and shows nothing of the great width which it carries on Silver Islet, nor have any of the rich silver minerals of that locality yet been found upon the mainland, or upon Burnt Island.

The experience gained in the working of Silver Islet Mine, since its discovery, has, in the main, confirmed the description here given of its geological relations. This will be seen in consulting the accompanying longitudinal section of the mine by Mr. W. M. Courtis, first published in the *Engineering and Mining Journal* of 21st December, 1878.

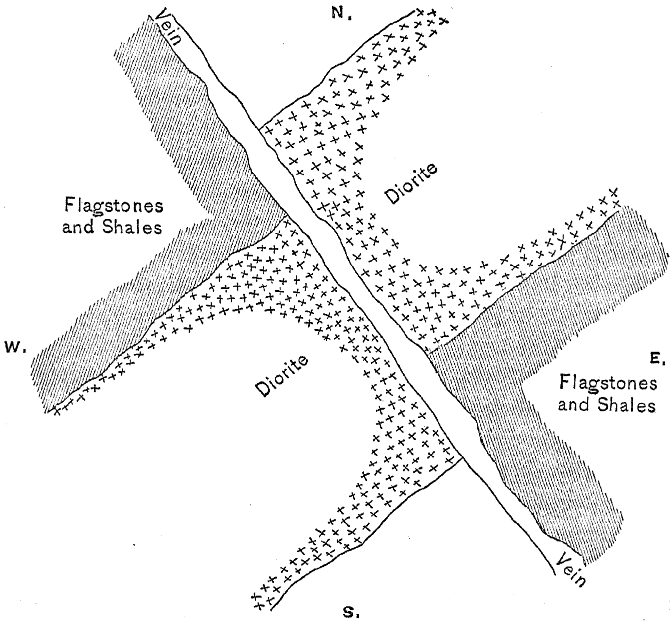
It will be observed that the diorite dike dips at a high angle southward, and that the workings have never been productive outside of it. Indeed, there are large areas of the vein inclosed by diorite walls on both sides which have yielded no ore. It is necessary



THE SILVER ISLET MINE, LONGITUDINAL SECTION.

to mention that the vein itself dips at a high angle to the eastward, and that Mr. Courtis's section shows the rocks adjoining the vein upon the hanging-wall only. On the west side of the vein it has been found that the diorite extends further south than on the hanging-wall. The line of the vein is, therefore, also the line of a fault,

which has moved the rocks on the west, or rather southwest, side of the vein eighty feet to the southeast, as shown in the accompanying sketch, which indicates roughly the relative positions of rocks and vein below the "5th level."



It is worthy of remark that the rocks intersected by the diorite dike are not highly inclined, semicrystallized slate, but are almost horizontal flagstones and shales.

The Silver Islet vein has exhibited a most remarkable phenomenon, which deserves mention here. The following is a description of it, taken from a letter dated 28th January, 1876, from Mr. W. B. Frue to the writer:

"On December 28th, while a party of miners were engaged in drilling a hole in the end of the drift on the '8th level,' the drill broke through into a small crevice or 'vug.' Water at once commenced to flow,—not in great quantity, however,—and one of the miners took a candle to look into the drill-hole, not being aware that there was a large escape of gas with the water. The gas instantly took fire, sending a flame out from the end of the drift for more than forty feet. The men, of course, threw themselves down on the bottom of the drift, and remained there uninjured until the flame subsided, and then went out to the shaft. After they had got over their fright, they

undertook to return into the drift, but when within forty feet of the end of the drift, the gas again ignited, filling the level with flame to within three feet of its bottom; the flame extending along the back of the drift, and burning for a distance of one hundred and fifty feet towards the shaft. The men again took shelter by throwing themselves down on the bottom of the level. Some time after this the men walked into and through the entire length of the drift without any light, and inserted a wooden plug in the hole through which the vapor and water were escaping. On the following day no gas was discovered in the drift, until a candle was brought close to the plug in the end of the level, when the gas again caught fire, giving a jet or flame about one foot long, which has been burning ever since."

This is, I believe, the first instance on record of the occurrence of an inflammable gas in a silver mine, and probably indicates that the rocks near Silver Islet are of much more recent age than has been generally supposed.

IV. Mining.

The discovery of Silver Islet in 1868, the full descriptions given of it, and the specimens of ore produced, valued at $1200, were insufficient to induce the Montreal Mining Company to go into vigorous mining operations. The leaning of the shareholders was plainly in favor of selling rather than working, and the largest specimens found their way to England, where they were exhibited, and afterwards sent to Swansea to be sold and smelted. Negotiations, having for their object the sale of part of Wood's Location, including Silver Islet, were carried on and continued into the summer of 1869, but unsuccessfully. Meanwhile I had returned to the lake, completed the survey of Wood's Location, and employed the men in excavating as much ore as possible from the surface of the Silver Islet Vein. This was a matter of much difficulty and even danger. The summer of 1869 was exceedingly stormy, and it was only during the calmest weather that any excavation was possible. The cartridges procured for blasting under water entirely failed to work, but, nevertheless, 9455 pounds of excellent ore were produced and shipped to Montreal. On the 12th of August we began to sink a shaft in the centre of the Islet, from which it was intended, on attaining sufficient depth, to cross-cut to the vein. That this plan would have proved perfectly successful is evident from the fact that it has since been sunk and connected with the east vein, and is now known as "Macfarlane's Shaft." During the fall of 1868 a shaft-house was built on the Islet, and a boarding-house, storehouse, and stable on the mainland, at a point now known as Silver Islet Land-

ing. In the winter of 1869–70 a party of twelve men and one horse remained on Wood's Location. They were instructed to continue sinking the shaft, to take advantage of any opportunity afforded by the formation of the ice to excavate more ore from the outcrop of the vein, and when work at the Islet was impossible to cut timber in the woods for the cribwork proposed to be constructed the following season.

The sinking of the shaft had to be discontinued on account of the influx of water, but the excavation of ore was very successful. The ice formed around the islet, and kept unbroken for nearly two months. It facilitated the work very much, keeping the water perfectly still, and affording the men a convenient platform. The blasting under water, with cartridges made on the spot by Mr. M'. Biggar, was successful, and extended over about thirty feet of the vein. The loose veinstone was then fished out of the water by means of long tongs, constructed on the spot, long-handled shovels, etc.; then teamed to the mainland, and there sorted. In this way nearly nine tons of ore were produced and shipped to Montreal in the spring of 1870. The following is a statement of the entire amount of ore produced during the operations of the Montreal Mining Company, with my assays and estimates of the value of the silver it contained:

When produced.	Net weight, lbs.	Percentage of silver.	Ounces per ton of 2240 lbs.	Value per ton of 2240 lbs.	Total value.
1868,	1,336	5.169	1,690	$2,095.00	$1,249.51
1869,	3,429	2.760	889	1,111.25	1,701.10
"	4,080	4.344	1,417	1,771.25	3,226.20
			Of 2000 lbs.	Of 2000 lbs.	
"	1,946	5.147	1,680	2,100.00	1,824 37
1870,	17,669	5.503	1,605	2,070.45	18,291.39
	28,460				$26,292.57

The above ore, after being sold or smelted, realized the following quantities and values of silver:

When sold or smelted.	Where sold or smelted.	Net weight, lbs.	Ounces per ton of 2240 lbs.	Value per ton of 2240 lbs.	Total value.
Sept. 4th, 1869,	Swansea,	1,209	1,397	$368⅕	$962.13
" "	New York,	127			190.50
" "	Swansea,	3,322	982	254$\frac{7}{11}$	1,821.96
Oct. 29th, 1869,	"	4,006	880	228¼	1,970.03
			Of 2000 lbs.	Of 2000 lbs.	
Feb. 24th, 1870,	Newark, N.J.,	1,913	1,608½	2,075	1,984.73
" "	"	2			11.28
Feb. 16th, 1872,	"	17,481	1,429	1,843.41	16,112.32
" "	"	13⅜			62.40
		28,073⅜			$23,115.35

It will be observed that this ore realized $3177.22 less than estimated, and that the greatest deficiency, in proportion to the quantity, occurs in the parcel of 4080 pounds sent to Swansea. No satisfactory explanation of the cause of the difference was ever received, and it was consequently deemed advisable to ship all the ore afterwards produced to Newark, N. J. The parcel sold there in February, 1870, resulted very satisfactorily; but a discrepancy occurred in the $8\frac{3}{4}$ tons afterwards sent. This parcel of ore, on its arrival in Newark, was crushed and sampled in the usual manner, and, as the first assays resulted much lower than that of the sample taken at Silver Islet, and given above, numerous trials were made by various New York assayers, always with most extraordinary differences as to yield. It is extremely probable that these differences were owing to the impossibility of effecting a perfectly equal distribution of the metallic grains throughout the mass of the sample. Mr. Balbach refused to account for more silver than his assay indicated, and efforts were made to effect a sale of the ore elsewhere than in Newark, but without effect, and it was finally treated by Mr. Balbach on the basis of his assay.

The facts connected with the production of ore in the winter of 1869–70 were published in the Montreal papers in June, and attracted much attention in England and the United States. That ten men had been able to produce $16,000 worth of ore, and that the actual time employed by them, in so doing, had not exceeded fourteen days, was again insufficient to induce the Montreal Mining Company to proceed to work the Silver Islet vein. The experience gained during the summer of 1869 had convinced me that very strong and extensive works would be necessary in order to protect the Islet and mine from the severe storms which frequently prevail off Thunder Cape. It had also become plain that a large force of men, with a steam-tug and scows, would be necessary for putting the cribbing into place, and securing it *quickly*, so as to prevent its being washed away while in process of erection by the heavy seas which rising gales suddenly bring in from the lake. I estimated that at least $50,000 would be required to establish a mine on the Islet, and recommended it as being for the interest of the shareholders that the company itself should work the mine.

In the event of their being unwilling to raise the necessary working capital, I recommended efforts on the part of the board to sell the Silver Islet property. Their decision resulted in favor of the latter policy, and not only was it decided to sell Silver Islet, but the

idea was adopted of endeavoring to effect a sale of the whole of the company's property, on the strength of the discoveries already made. Negotiations were carried on for this purpose during the spring and summer of 1870, which resulted in the transfer of the whole property into the hands of certain capitalists in New York and Detroit in September.

The first intimation which I had of the sale of the whole property was at Silver Islet on the night of the 31st of August, when the propeller "City of Detroit" arrived, having on board Mr. W. B. Frue, a working party of about thirty men, two horses, machinery, stores, provisions, etc., and having in tow a large scow and a raft of large-sized timber. The propeller discharged her cargo next day, and operations were begun at once and vigorously to establish a permanent mine on Silver Islet. In spite of severe weather extensive breakwaters were built, part of the vein inclosed by a coffer-dam, the area within the latter pumped dry, a considerable amount of mining done, and about seventy-seven tons of ore shipped before the close of navigation. The time devoted to mining was about four weeks, and the last shipment was made about the 25th of November. About $80,000 were expended in the above operations, and in making provision for wintering, long before any returns were obtained from the ore. Indeed, it was not until March, 1871, that the smelting of the fall shipments at Newark was completed. I was employed by the new company to superintend the sampling of this ore, and the following statement shows the value of the silver in the ore produced by them in 1870, according to my assays:

No. of parcel.	Net weight, lbs.	Percentage, silver.	Ounces per ton of 2000 lbs.	Value per ton of 2000 lbs.	Total value
1	34,862	2 553	744.8	$960.90	$16,749.53
2	16,592	2.156	628.8	811.15	6,729.30
3	17,523	2.690	784.5	1,012 00	8,866.63
4	17,772	2.973	867.2	1,118 75	9,941.34
5	16,379	3 358	979.4	1,263.48	10,347.26
6	15,914	3 306	964.2	1,243 81	9,896.99
7	19,139	4.348	1,268 1	1,635 84	15,654.17
8	17,139	4.687	1,367.0	1,763.43	15,111 71
Metallic grains.	33	15 136	4,414 6	5,694.83	93.96
Specimens.	187	9.060	2,652.4	3,421.59	319.92
Metallic grains.	3	67.658	19,733.3	25,455.95	38.18
	155,543 lbs., averaging per ton			$1,205.44	$93,748.99

Messrs. Balbach's assays of the same ore are given in the subjoined table:

No. of parcel.	Net weight, lbs.	Percentage, silver.	Ounces per ton.	Value per ton.	Total ounces.	Total value.
1	34,862	2.406	701 9	$905.46	12,234.9	$15,783 15
2	16,592	2.250	656.0	846.24	5,442.1	7,020 40
3	17,523	2.680	781.6	1,008.26	6,847.9	8,833.86
4	17,772	2.905	847 5	1,093.28	7,531.0	9,715.00
5	16,379	3.385	987.3	1,273 63	8,085.4	10,430.39
6	15,914	3.170	924 5	1,192.60	7,356.2	9,489.51
7	19,139	4.220	1,230.0	1,586.70	11,770.4	15,183.92
8	17,139	4.510	1,315 0	1,696 35	11,268 8	14,536.87
Metallic grains.	33	15.136	4,414.6	5,694.83	72.8	93.96
Specimens.	187	9.060	2,652 4	3,421.59	248 0	319.92
Metallic grains.	3	67.658	19,733.3	25,455.95	29.6	38.18
	155,543,	averaging	per ton	$1,175.80	70,887.1	$91,445.16

While this ore was being disposed of in the winter of 1870-71 Mr. Frue and his men were fully occupied at Silver Islet, where a very stormy season was experienced. During the previous winter the ice had formed quietly and remained till the spring, when it was gradually softened by the heat, moved out quietly into the lake, and did not reappear. Mr. Frue was, therefore, somewhat unprepared for the great trouble which the different behavior of the ice during the following season caused him. The placing of the cribbing was tedious and hazardous work. The ice did not form solidly for any considerable time, but kept floating backwards and forwards the entire season. During much of the time Mr. Frue and his men had to cut their way from the mainland to the Islet, through fields of solid ice recently formed, and, shortly afterwards, through floating floes three feet in thickness and of all sizes up to fifty feet square. Towards the end of February severe gales were experienced, which lent tremendous force to the floating ice, and tore away cribwork to an extent of two hundred and forty feet in length. The heaviest timber was insufficient to withstand the ice; large logs had their extremities chafed to such an extent as to resemble only huge brooms, and bolts, one and a half to two inches in diameter, were twisted and broken apparently with the greatest ease. After the removal of the cribbing the seas were so heavy as to dash over what remained, and fill up the coffer-dam in a very short time. The dam itself, however, sustained very little damage, and, going to work indefatigably to repair damages, Mr. Frue was able to resume mining in about a month afterwards. By the 1st of May, 1871, an excavation had been made on the rich part of the vein inclosed by the coffer-dam, leaving a length of sixty-five, depth of thirty-two, and an average width of eight feet. By the close of navigation, in November, this working

had attained a depth of ninety feet, and had produced, from the same time in 1870, about four hundred and eighty-five tons of ore, of which the following quantities were treated at Newark in the summer of 1871:

No. of parcel.	Net weight, lbs.	Ounces per ton of 2000 lbs.	Value per ton of 2000 lbs.	Total ounces.	Total value.
1, A.	6,553	1,184.1	$1,539.33	3,879.70	$5,043.61
1, B.	14,986	1,406.4	1,828.32	10,538.15	13,699.60
2, A.	16,627	1,149.1	1,493 83	9,553.04	12,418.95
2, B.	3,824	1,470.1	1,911.13	2,810.83	3,654.08
3	17,393	1,131.6	1,471.08	9,840.95	12,793.23
4	19,651	1,166 0	1,515.80	11,456 53	14,893.48
5	18,630	1,281.8	1,666.34	11,939.96	15,521.95
Metallic grains.	375	4,145.0	5,388 50	772.20	1,003 86
6	17,959	1,198 7	1,558 31	10,763.72	13,992.85
7	16,730	1,134.5	1,474.85	9,490.09	12,337.12
8	17,954	1,105 3	1,436.89	9,894.64	12,863.03
9	18,975	947.8	1,232.14	8,992.25	11,689 92
10	13,189	907.0	1,179.10	5,981 21	7,775 57
Metallic grains.	657	1,415.8	1,840.54	465.10	604 63
	183,453			106,378.37	$138,291.88

There were further treated, at the Wyandotte Smelting and Refining Works, the following parcels of ore, and in the order as they are here given:

Quality of ore.	Net weight, lbs.	Ounces per ton of 2000 lbs.	Value per ton of 2000 lbs.	Total ounces.	Total value.
No. 1.	6,919½	4,180.0	$5,434.00	14,462.00	$18,800 60
No. 2.	16,918	1,395.0	1,813.50	11,799.00	15,338.70
"	19,340	1,139.0	1,480.70	11,014.00	14,318 20
"	7,500	1,332.0	1,641.60	4,995.00	6,493.50
"	13,750	1,318.0	1,713.40	9,061.00	11,779.30
"	49,500	961.8	1.250.34	23,804.55	30,945.91
"	38,750	1,671.4	2,172.82	32,383.37	42,098.38
"	36,750	1,077.3	1,400.49	19,795.38	25,733.99
"	60,000	1,258.4	1,635.92	37,752.00	49,077 60
"	64,343	1,080.0	1,404.00	34.745.00	45,168.50
"	4,000	990.9	1,288.17	1,981.80	2,576.34
"	55,000	997.4	1,296.62	27,428.50	35,657.05
No. 3.	15,000	183.2	238.16	1,374.00	1,786 20
"	20,000	108 6	141.18	1,086.00	1,411.80
No. 2.	16,232	997.4	1,296.62	8,094.89	10,523 36
"	20,253	615.0	799.50	6,227.79	8,096.13
"	15,607	270.6	351.78	2,111.62	2,745.10
"	27,017	990.9	1,288.17	13,385.57	17,401.24
"	34,469	1,107.0	1,439.10	19,078.59	24,802.17
"	37,615	925.5	1,203.15	17,406.34	22,628.25
"	37,500	545.0	708.50	10,218.75	13,284.38

Quality of ore.	Net weight, lbs.	Ounces per ton of 2000 lbs.	Value per ton of 2000 lbs.	Total ounces.	Total value.
No. 2.	18,560	1,029.0	$1,337.70	9,549.12	$12,413.86
"	24 605	754.0	980.20	9,276.08	12,058.90
"	17,998	814.3	1,057.59	7,327 88	9,526.24
"	17,002	680.3	884.39	5,783.23	7,518.20
"	15,929	813.5	1,057.55	6,479.12	8,422.86
No. 3.	17,500	144.8	188 24	1,266.50	1,646.45
"	35,461	137.3	178.49	2,434.39	3,164.70
No. 1.	10,483	3,617.2	4,702.36	18,959.55	24,647.41
"	6,495	3,617.2	4,702.36	11,746.85	15,270.90
No. 2.	2,982	1,361.2	1,769.56	2,029.54	2,638.40
"	14.990	684.1	889.33	5,127.32	6,665.51
	778,468½			388,184.73	$504,640.13

It is, of course, to be remembered that the values just given do not represent the amount realized for the ore. Both at Newark and Wyandotte the smelters only guaranteed to return 95 per cent. of the silver contents, and charged $100 per ton for smelting. Besides the above ore there was produced in 1870-71 another parcel of five tons, which, with many lives, was lost on board the propeller Coburn, on Lake Huron, in October, 1871.

The total production of Silver Islet from the discovery till the close of navigation, 1871, was as follows:

	Weight, lbs.	Value per ton.	Total value.
Under Montreal Mining Company, . .	27,073⅝	$1,646 80	$23,115.35
Under new proprietors, 1870, ...	155,543	1,175.80	92,153.23
Under new proprietors, 1871, Newark,	183,453	1,507.64	138,291.88
Under new proprietors, 1871, Wyandotte,	778,468½	1,296 48	504,640.13
Lost on propeller Coburn, ...	10,000	1,040.00	5,200.00
	1,154,537⅛	$1,322.44	$763,400.59

Mining was continued with varying success after the close of navigation in 1871. The vein was found to be subject to frequent and sudden changes, both as regards size and richness. In the fall of 1871 it narrowed down to six inches in width at some points, with scarcely any first quality ore in sight. During the winter it gradually widened and became very productive. In Mr. Frue's reports many such alternations are recorded. He says that in the summer of 1872 "the lode became broken up, being thoroughly mixed with diorite and wedges of plumbago, and in the fall the mine assumed anything but a flattering appearance." Mr. Frue writes further on this subject as follows: "In the following winter it suddenly changed in character and produced, up to May 1st, 1873, 250 tons of rich

packing ore, worth about $1500 to the ton. During May and the early summer the vein disappeared almost entirely, being broken up into strings and feeders. Later, however, there was a decided improvement, which was again overshadowed by a passing cloud, and although in extending the drift north on the forty a very promising show of silver had been opened, I had often seen the mine clothed in richer apparel than it appeared in at the close of navigation" (1873).

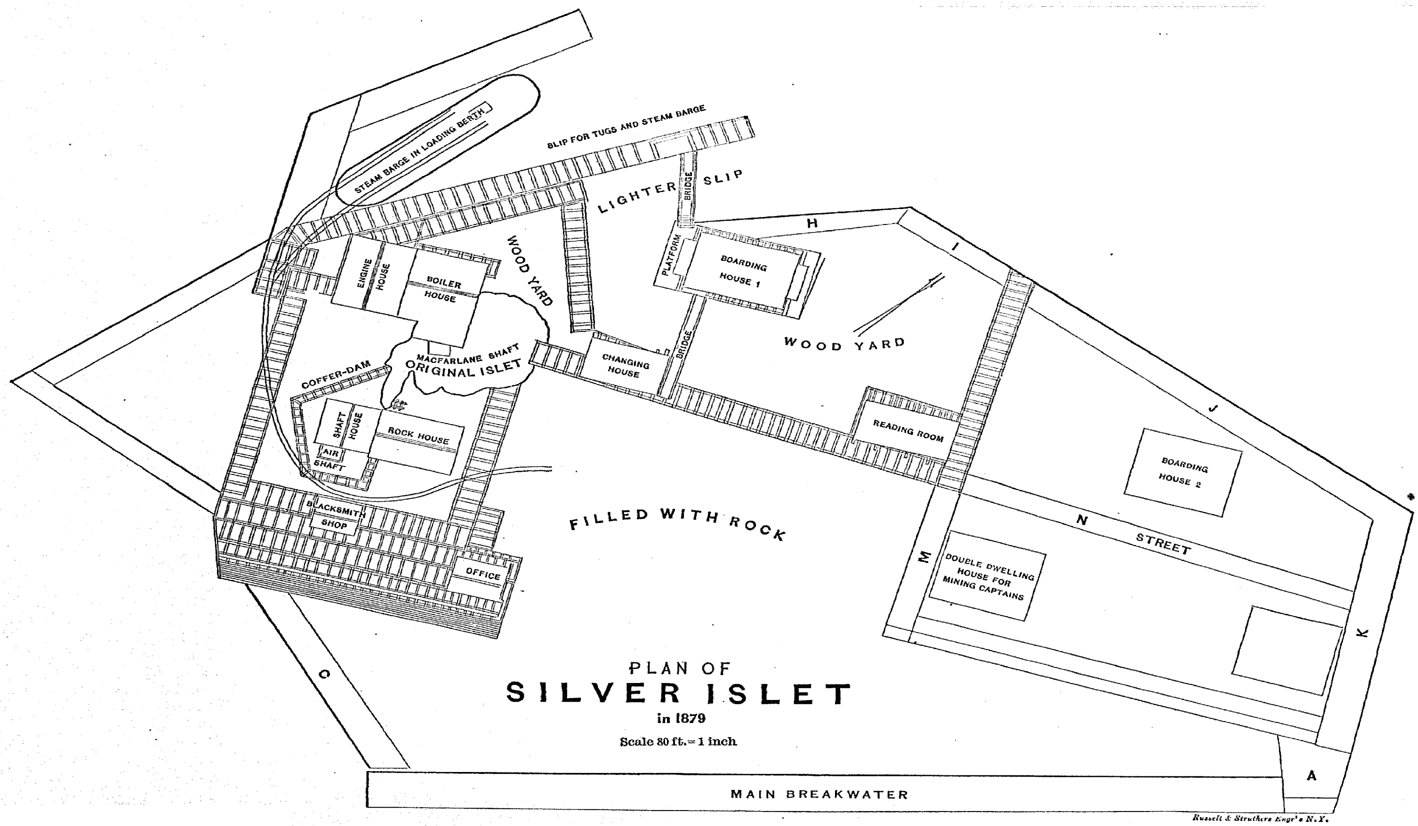
The severe storms which marked the winter of 1870–71 put in an appearance again in that of 1873–74. "About the middle of November," Mr. Frue reports, "we were visited by a heavy storm from the southeast, which did considerable damage, amounting in all to about $2000. On December 1st we were again overtaken by a southeaster which came on in terrible fury and seemed, for a time, as though it would sweep everything before it. It tore away nearly 350 feet in length of submerged cribs, and caused a loss of 20,000 feet of timber, 7½ tons of bolts, and nearly 5000 tons of rock" (used in loading the cribs), "the total destruction amounting to a little over $9000, besides the carrying away of the upper portion of the main breakwater. This work had an altitude of nearly twenty feet above the level of the lake. Eight feet in height of the top, and nearly sixty feet in length of it was carried away, the breach being directly in the centre. The blacksmith's shop, which stood inside of this breakwater and about forty feet from its outer face, was completely demolished. In fact, rocks were whirled around the Islet like hailstones, and a number of buildings were damaged to a considerable extent." The situation of the works protecting the mine will be seen from the accompanying plan of Silver Islet in 1879.

The ore produced from 1874 to 1875, inclusive, was treated at the Wyandotte Works and contained the following quantities of silver:

Season of 1872, ...	310,744 02 ouncss.
" 1873, ...	289,763.77 "
" 1874, ...	250,021.75 "
" 1875, ...	145,902.50 "
	996,432.04 ounces.

At $1.20 per ounce the value of this product amounts to $1,195,718.45.

Part of the product of 1875 was in the shape of concentrates from the stamp mill, which had been built at a cost of $90,000 for treating the veinstone of inferior quality which had previously been laid aside


PLAN OF
SILVER ISLET
in 1879
Scale 80 ft. = 1 inch
STEAM BARGE IN LOADING BERTH
SLIP FOR TUGS AND STEAM BARGE
LIGHTER SLIP
BRIDGE
PLATFORM
BOARDING HOUSE 1
WOOD YARD
WOOD YARD
ENGINE HOUSE
BOILER HOUSE
MACFARLANE SHAFT
ORIGINAL ISLET
COFFER-DAM
SHAFT HOUSE
AIR SHAFT
ROCK HOUSE
CHANGING HOUSE
BRIDGE
READING ROOM
BLACKSMITH SHOP
OFFICE
FILLED WITH ROCK
BOARDING HOUSE 2
DOUBLE DWELLING HOUSE FOR MINING CAPTAINS
N STREET
H
I
J
K
M
C
A
MAIN BREAKWATER
Russell & Struthers Engr's N.Y.

as too poor for shipment. This mill has 50 stamps and 12 Frue vanning machines, and produced about 1 ton of concentrates from 50 tons of the poor veinstone. The following statement gives the quantity of this ore stamped, and the concentrate produced from May, 1875, to November, 1876:

Month.	Tons rock stamped.	Product in concentrates. Tons. Lbs.	Total ounces of silver contained in same.	Total cost of dressing.
May, 1875,	541	11.1454	10,210	$1,237.69
June, "	1,065	25. 212	17,552	2,049.89
July, "	1,079	28. 104	19,125	2,427.33
August, "	762	20.1100	11,238	2,302 78
September, "	1,505	35. 182	17,804	2,990.85
October, "	1,678	37. 843	14,415	2,840.15
December, "	1,642	31. 847	11,548	3,172 26
January, 1876,	1,556	30.1824	15,990	3,089.96
February, "	1,421	28.1312	16,346	2,944 89
March, "	1,690	32. 651	15,754	3,238.60
April, "	645	10.1475	4,806	1,788.83
May, "	1,673	33. 583	9,614	3,036.62
June, "	1,565	33.1288	10,504	2,891 36
July, "	1,525	41. 978	11,757	2,768.47
August, "	1,600	39.1591	9,527	2,863 04
September, "	1,505	33.1232	10,060	2,807.17
October, "	1,500	38.1835	9,234	2,596.26
November, "	1,494	29. 194	11,389	3,093.93
	24,446	$541\frac{1705}{2000}$	226,873	$48,145 08

These figures show that 9.28 ounces per ton were extracted from the stamp rock and concentrated in a product containing 418 ounces per ton, at a cost of $1.97 on the original ore.

The unfavorable changes in the Silver Islet vein, which occurred in the fall of 1873, continued up to the close of 1876. The new levels which had been opened up, the eighth and ninth, proved wholly unproductive, although no difficulty was experienced in following and working on the vein. A vast amount of exploratory work by means of the diamond drill also failed to discover any deposits of rich ore. The consequence was, of course, great financial embarrassment and an almost entire cessation of work during the summer of 1877. In August of that year work was resumed, and up to December 23,850 ounces of silver obtained by stoping in the upper part of the mine. It was even proposed to remove the rich ground lying betwixt the mine and the lake, substituting for it an artificial arch; but, fortunately, in the summer of last year a bunch of rich ore was struck beneath the fourth level, south of the shaft, which in a few months yielded 721,632 ounces of silver, a quantity amply

sufficient to rescue the mine from all its embarrassments and provide a reserve or working capital of $300,000.

I have not found it possible to ascertain the amount of the product, year by year, subsequent to 1875, but according to information received from C. A. Trowbridge, Esq., Secretary of the Silver Islet Company, there have been extracted since the commencement of operations, in September, 1870, and up to the close of navigation in 1878, 2,174,499½ ounces refined silver, with a value of $2,921,727.24. If to this we add the value of the ore obtained immediately after the discovery by the Montreal Mining Company, we have a total yield of $2,948,019.81. With regard to the production of the present year, it consists almost exclusively of concentrates, but its value is very certain to exceed $50,000, and I think that at the end of the season the total yield of Silver Islet mine since its discovery will be found to have reached three million dollars.*

The question of the future of Silver Islet mine is one of very great interest to many besides those pecuniarily concerned. When visiting the mine, in July, 1877, the vein appeared perfectly well defined on the ninth level, but nothing in the shape of ore was to be seen. The vein was said to possess the same character in the inclined shaft sunk 100 feet deeper than the level, and to a point about 640 feet from the surface. This shaft was filled with water at the time of my visit. The vein below this point has been tested by a drill-hole 296 feet deep, in which traces of silver ore were detected. Even if we suppose that this trace is the clue to another *bonanza,* the fact still remains that from the 6th level to the deepest working, a distance of 300 feet, the vein has been found to be unworthy of excavation, and this too in spite of the presence of diorite on both walls, a condition which, when the mine was first opened, was supposed to insure a remunerative vein. In view of this fact and of the circumstance that the recent rich discovery was made at a point where only the foot-wall could have been diorite, it becomes a question whether the theory of the beneficial influence of diorite walls is correct. If it is, then a large amount of vein area below the 5th level and to the north of the shaft remains to be prospected. That this ground has, so far, been found barren may be owing to the peculiar nature of the vein, in which large values of ore seem to be stowed away in comparatively small space. If the theory here referred to is unfounded, and the vein in the adjoining flags and shales be really as promising as that crossing the diorite, then the amount of ground available for

* I have learned from Mr. Trowbridge, since reading this paper, that the precise yield was $3,039,557.49.

exploration, north and south of the mine, is immense. As a matter of fact, the horizontal strata, elsewhere in the district, have been found to contain silver-bearing veins, but, so far, have failed to afford foundation for a remunerative mine. In either case the future of Silver Islet mine depends mostly on the carefulness of the manager and his assistants in detecting minute traces of ore, and their skill and perseverance in following them. It would seem altogether unwise to depend upon any preconceived notion as to the manner in which the valuable minerals occur, or ought to occur, in the mine, for it must be confessed that hitherto the chemical geologist has rendered but very slight assistance to the practical miner in his search for the remunerative parts of a vein. The best guarantee for the future is in the past history of the mine, which proves that rich deposits may be stumbled upon quite unexpectedly in the ground already opened up.

I obtained from Edward Learned, Esq., President of the Silver Islet Company, whose faith and perseverance has carried the company through many of its difficulties, the original of the panoramic sketch, published herewith, of Silver Islet and the adjacent islands and mainland. It is exceedingly truthful, and will serve to give some idea of the situation of this remarkable mine. The artist is Mr. George Snell, of Boston.

I have thus endeavored to record the principal facts which have come to my knowledge regarding this extraordinary silver vein. Its story ought to teach Canadians, among other things, to have more confidence in the mineral resources of their country. That over three millions have been extracted from a bare rock, in Lake Superior, with an area not exceeding a thousand square feet, ought to increase our faith in the vast unexplored regions which stretch away to the north and northwest of us. But let us not, in the event of new discoveries, pamper our worthless mines, nor, on the other hand, starve those of good promise. Neither let us, when we find another productive mine, tear out recklessly all the ore in sight. The product of a mine, like that of a farm, cannot be forced beyond certain proper limits without bad consequences. Let reserves accumulate in our mines as the "rests" formerly did in our financial institutions, and mining will become as profitable as banking, if not more so. The opposite system, "picking the eyes out of the mine," *Raub bau*, as the Germans call it, has caused the ruin of such mines as the Ophir in Nova Scotia, and the Acton in Quebec. It is "more by good luck than good guiding" that a similar fate has not yet overtaken Silver Islet in Ontario.

A NEW METHOD OF DREDGING, APPLICABLE TO SOME KINDS OF MINING OPERATIONS.

BY ROSSITER W. RAYMOND, PH.D., NEW YORK CITY.

I DESIRE to call the attention of the Institute to a novel system of dredging, which, it seems to me, may prove applicable, not only to river and harbor improvements, but also to certain varieties at least of alluvial and diluvial gold mining; that is, river-mining, bar-mining, coast-mining, and any other similar operations now carried on by means of coffer-dams or dredging machines.

The simple and effective method to which I refer is the invention of General Roy Stone, formerly an officer of volunteers, and at present a civil engineer, engaged under General Newton, of the United States Engineer Corps, upon the government works in New York harbor. General Stone has been, since September, 1878, in charge of operations upon Diamond Reef. This reef, lying between Governor's Island and the Battery, is one of the most dangerous in the harbor. Its position in the highway of navigation, and the fact that it is dangerous only to vessels of considerable draught and at certain stages of the tide, combine to increase the serious nature of the disasters which it has caused. The reef consists partly of rock in place, and partly of deposits of glacial clay, boulders, and pebbles, over all of which lies, or did lie until it was removed, the ordinary silt and rubbish of the harbor. As the purpose of the present paper has nothing to do with the special dimensions and other circumstances of this particular work, I describe those features only which are connected with General Stone's ingenious device.

Various attempts have been made to remove the upper portion of Diamond Reef and secure a navigable depth at low tide. Some years ago Maillefert undertook to accomplish this by firing charges of high explosives, simply laid upon the surface. He accomplished some good in the removal of projecting points and ridges; but as soon as the surface of the reef had become approximately level, his method was no longer effective.

After the work had been taken up by the Engineer Corps, the well-known drilling-scow invented by General Newton was brought into action; and, so long as holes could be bored and fired in solid rock, this process was effective enough. In the mixture of hard-pan

and pebbles, however, boring proved impracticable. The suggestion of a pile-driver, to drive piles, which could be subsequently withdrawn to leave holes for blasting, appears to have been made by one of the officers employed, but never to have been carried out. Under the circumstances, I should hardly consider this plan worth a trial. Even if it could be executed, the most satisfactory results which could be expected from it would fall far short of the simple, rapid, and cheap performances of Stone's apparatus.

By a thorough examination, General Stone found the whole upper portion of the reef to possess apparently one and the same character, that of an exceedingly compact indurated clay (hard-pan), filled with boulders, and containing occasional pockets of gravel and sand. The size of the boulders indicated that no bed-rock existed on this part of the reef, within the depth required for navigation. The clay was too hard to be removed by ordinary dredging, and, after some consideration, it was resolved to try the effect of a sort of hydraulic mining; that is, of the use of powerful streams of water from a force-pump. A strong Worthington pump was mounted on the scow, and divers were sent down with the hose-pipe, but they were unable to hold it against the reaction of the stream.

The next day, the pipe being lashed to a pole held from the deck and guided only by the divers, it was found that the hardest clay was rapidly penetrated and the earth and small stones were washed away. The effect of the streams appeared to be fully equal to that which they would have produced if directed against the same material on land. As the result of this successful experiment, simple means were devised to control and guide from the deck the hose and their nozzles at a depth of 30 feet, and in the most rapid tidal currents, when the divers could not remain below. The hose-pipes finally employed are of heavy iron, 2½ inches in diameter, and 12 to 16 feet long, contracted to 1⅛ inch at the nozzle. These preserve the hose from contact with rocks on the bottom. At 3 feet from the nozzle, they are attached firmly to the ends of heavy spars, provided with steam tackle for lifting and lowering, and various other guys, braces, and tackle to hold the hose in position against the tide and move it as required, to hold it down against the reaction of the stream itself, and to rock and twist it so that the nozzle may work its way down among the boulders.

Five or six men are required to each spar, and one at the pump. A pressure of 150 pounds to the square inch is used. This will pene-

trate the level surface of hard-pan at the rate of one foot per minute, making a "pot" from three to five feet in diameter, in which boulders of 20 pounds weight boil up and remain suspended until the stream is withdrawn. On a slope, it is still more effective, and in sand the pipe sinks almost as if in water. The accumulation of larger stones in the hole usually stops the descent of the pipe at about 5 feet, the spar being too large to penetrate among them when they are jammed in the smaller section at the bottom. On face-work, a somewhat greater depth may be reached, but the stones roll to the foot of the bank and impede the pipe and spar there so that it is necessary to rake them away. This is done by the divers at slack water, and at other times with a long-handled rake guyed against the tide, and guided by hand and hauled forth and back by steam-tackle. The rake is worked by about the same number of men as a stream, and is fairly effective when it does not encounter rock too large for it to move.

Beyond the fact thus demonstrated, of the effectiveness of a subaqueous jet, the proceedings of General Stone up to this point have no special bearing upon mining operations, in which it is required that the excavated material shall be brought to the surface for further treatment. The first operations according to the method above described washed the sand and earth from the edges of the reef away into deep water, leaving the larger stones to be removed by grappling; but as soon as a flat surface had been formed by the cutting and filling, the material began to lodge and to return at the next tide. It was evidently necessary to devise a method of carrying it athwart the tide into the deep water alongside the reef, whence it could not return.

The customary methods of dredging and dumping, whether with the ordinary dredges or by pumping, seemed extravagant when the material needed to be moved so short a distance and not to come to the surface at all. In lieu of these it, was suggested to lay a pipe along the bottom and throw a jet of steam into it to create an induced current (as in the steam siphon) sufficient to move the gravel and sand; but this would involve a considerable loss by radiation in the passage of the steam, however well it might be protected, and a very great loss by condensation when the steam strikes the water in the pipe.

The only remaining method which occurred to General Stone as practicable was to create an induced current by water instead of

steam, a method for which the scow was already provided with the necessary appliances, except the large pipe for the bottom.

Before obtaining a large pipe, General Stone experimented with the drill-pipes on the scow. Into a vertical pipe 6 inches in diameter, suspended in the water so as to leave 2 feet of water standing in it, a $\frac{5}{8}$-inch stream under a pressure of 150 pounds per square inch was thrown from beneath. When the nozzle was placed 6 inches below the bottom of the pipe no effect was seen. When placed at the bottom, the water in the pipe was raised $3\frac{1}{2}$ feet; placed 10 inches above the bottom, it raised the water $4\frac{1}{2}$ feet and the injected stream did not penetrate the 14 inches of water above it so as to rise above the surface.

A pipe with bell-mouth was then sent down and laid horizontally on the bottom. A $\frac{5}{8}$-inch nozzle was inserted one foot beyond the bell-mouth, and one of the divers used a similar stream to stir up the sand and stones near the bell-mouth, while the other observed the stream emerging from the pipe and endeavored to get samples of it in strong glass jars. But the current swept him away from the pipe, and stones as large as his fist came through with such force as to bruise his hands. He came up with his hands bleeding and his jars broken; and the other diver confirmed his statement of the force of the stream by saying that his arm had been drawn into the mouth of the pipe with great force.

The pipe was then brought near the surface, and the stream injected with intent to measure the velocity of the current through the pipe, but the length of the pipe (10 feet) was not sufficient to get any appreciable space of time in the passage of an object through it.

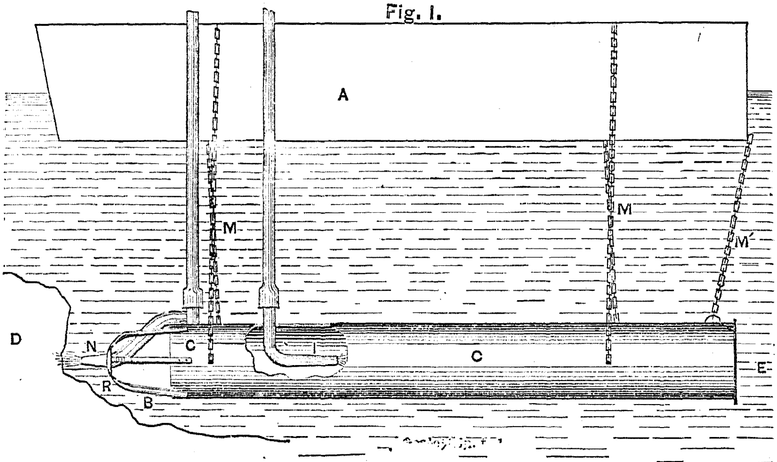
When the large pipe, 15 inches diameter and 64 feet long, was received, a successful measurement of velocity was made. The injected stream was through a $1\frac{1}{4}$-inch nozzle, under pressure of 150 pounds, with a volume of about 400 gallons per minute. The velocity in the large pipe was 10 feet per second, giving a discharge of 5500 gallons per minute, from which, deducting the injected 400, leaves 5100 gallons per minute as the volume of induced stream.

For the foregoing facts I am indebted to the courtesy of General Stone, who, besides answering fully my inquiries, placed at my disposal his official report of progress. The important question, how much solid matter can be transported by the induced stream, is not yet precisely answered. In the operations at Diamond Reef, from 24 to 32 feet under water, the pipe removes all that the stirring-jet outside can dig up, and it is the latter which limits the capacity of

the whole apparatus. The stream there carries from 2 to 5 per cent. of its volume in the form of solid material. But when the material to be removed is already loose, such as sand or gravel, and does not need to be bored into and dug up by a hydraulic jet, the induced current seems to carry 20 to 25 per cent. A basket or coarse grating over the mouth of the pipe prevents the entrance of boulders of excessive size. But stones larger than one's fist are frequently carried through.

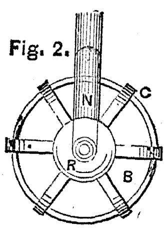
In the illustrations herewith given, Figs. 1 and 2, the large pipe *C* is shown suspended by chains *M M′* from the scow *A*. A basket-

Fig. 1.



grating *B* prevents the entrance of large boulders into the pipe. *I* is the interior or propelling, and *N* the exterior or excavating jet, the latter being held in its place by the disk *R*, through which it passes, and which forms the centre-piece of the basket. The pipe *C* is represented as horizontal, and the nozzle *N* as pointing backward, directly in the axis of the pipe. This would be the proper position for both, if the material to be excavated were standing as at *D*, and the object were to remove it horizontally and drop it into deeper water at *E*. But the direction given to the nozzle *N* may be varied

Fig. 2.



Scale ¾ in. to 1 ft.

according to the surface attacked, against which, for maximum effect, it should deliver its jet at a right angle. For instance, the pipe *C* being horizontal, and the surface of attack also horizontal, the nozzle *N* would be directed vertically downward. It is, however, in most

cases best to incline the pipe *C*, so that its discharge may be above the surface of the water and continually open to inspection. In the apparatus now employed at Diamond Reef the pipe *C* is thus inclined, so that its lower end touches the reef, while its upper end projects a foot or 18 inches above tide, and can be observed from the deck of the scow. It is by watching the discharge, and noting the quantity of mud, sand, and stones which it carries, that the workmen know when the jet *N* "takes hold" (usually about ten seconds after operations commence at a given spot), and when it ceases to be effective, either through obstruction in the pipes, or through the accumulation of stones before the nozzle. In the former case, the lower end of the pipe is easily brought to the surface for examination; in the latter case, it is moved a short distance under water, to commence operations at another spot, and dredging is not resumed at the first spot until the stones have been removed by raking, grappling, etc. These operations are controlled by the periodical inspection made by the divers.

This form of the apparatus seems likely to be very useful in some kinds of gold mining. Since an interior jet of 1¼ inch diameter under a pressure of 150 pounds per square inch will carry a large column of water at least a foot above the surface, it is evident that in reasonably still water there would be no difficulty in causing the large pipe to discharge its contents into sluices floating upon rafts, or, even under some circumstances, into sluices set along the shore of a river. Increasing the pressure of the jet, or diminishing the size of the larger pipe, would increase the height to which the discharge might be raised.

On the coast of Oregon and Northern California there are, as is well known, deposits of auriferous sands, which exhibit, when washed up by the tide, placers of sufficient richness to repay mining. It is true, that these sands probably have their origin in bluffs on the shore which have been gradually eaten away by the ocean; and it is questionable whether dredging under water along this coast would bring up material of sufficient richness to be profitably worked. But if such be the case, an apparatus of the kind I have described would be a very simple and effective means of raising the sands and loading them upon suitable barges.

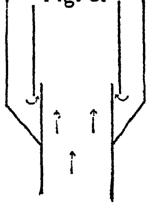
A feature which particularly recommends this device for experimental use is the circumstance that it involves little loss in case of failure. The Worthington or other high-pressure steam-pump and boiler, the scow and tackle, and even the pipe and hose, are all useful

and salable for other purposes; so that a machine of this kind would be by no means a dead loss if it should not in a given locality raise, from a river or ocean bottom, material rich enough for profitable working. The whole apparatus could either be towed or floated to another locality for further experiment, or it could be broken up and the parts sold separately for other uses.

I do not anticipate that the field in gold mining, for an apparatus of this kind, will be very large. The use of General Stone's contrivance for dredging of rivers, harbors, canals, and the like will be far more important and extensive; but its simplicity, cheapness, portability, and effectiveness, as well as its novelty, seem to me to justify calling to it the attention of such mining engineers as may have to deal with the problems to which it is applicable.

Concerning the novelty of this use of water, it ought to be noted that Captain Eads has employed, at the mouth of the Mississippi, a similar device in dredging. I have not seen either his contrivance or precise drawings of it. I understand, however, that in that case the water under pressure is forced through an annular space into the dredging-pipe, somewhat as the accompanying imaginary diagram (Fig. 3) indicates. It seems to me that the central jet, as used by General Stone, would be more effective, since it would deliver the same amount of water with less friction; the friction for a circular opening being less than for an annular opening of the same area. Whether there may not also be an advantage in the transmission of power from a central jet, as compared with the circumferential one, to the main column of water to be lifted, and whether, moreover, any difference in the effectiveness between the two plans could be great enough to make itself felt in practice, I am not prepared to say. At all events General Stone's arrangement appears to be, for ordinary purposes, the simpler, and to have no serious defects.

Fig. 3.



I believe this invention has been made the subject of an application for a United States patent, and that the Patent Office having reported favorably upon it, the patent will shortly issue to General Stone.

THE NEW RIVER COAL-FIELD OF WEST VIRGINIA.

BY S. FISHER MORRIS, M.E., QUINNIMONT, W. VA.

THE New River coal-field embraces that portion of the Appalachian coal formation which lies on the waters of the New River, principally in Fayette and Raleigh counties, West Virginia, covering a strip of territory about forty miles in length along the line of the Chesapeake and Ohio Railroad from Quinnimont to Kanawha Falls, where the New River empties into the Kanawha, the railroad following the banks of the river the whole distance. (See map.)

Between Kanawha Falls and the Valley of the Ohio lies the great Kanawha region, and these two coal-fields possess a greater variety of coal than can be found in any other portion of the country, enabling them to furnish the best fuels for any of the various demands of manufacture or domestic use, the Kanawha region possessing many varieties of gas, hard splint, and cannel coal, and the New River region bituminous and semi-bituminous, steam and coking coal.

The New River, in its westward course towards the Kanawha, has cut its way entirely through the Seral conglomerate or No. XII of the Pennsylvania Reports, the Umbral red shales of No. XI appearing at the foot of the mountains at the eastern end of the region at Quinnimont, one of the upper ledges of the Conglomerate forming the summits; while at the western end, at Kanawha Falls, the top of the Conglomerate is nearly down to the level of the river, a portion of the "Lower Productive Measures" forming the greater portion of the mountains.

These mountains rise abruptly from the banks of the river to a height of from 800 to 1200 feet, leaving only here and there narrow strips of bottom land, a few acres in extent; and as the measures consist principally of hard sandstones, shales, and conglomerates lying in a nearly horizontal position, the mountain-sides are very precipitous, with many long and high cliffs, which, with the great height of the mountains, give the country a very rugged appearance.

Along the sides of these mountains fronting on the New River and its many tributaries there are exposed the outcroppings of several veins of bituminous and semi-bituminous coal, varying in thickness from a few inches to over four feet, five of them being workable, or

containing three feet of coal and upwards. Of these seams only two appear to crop out with a workable thickness at or near the river front, the highest seams being in the high hills a short distance from the cañon of the New River.

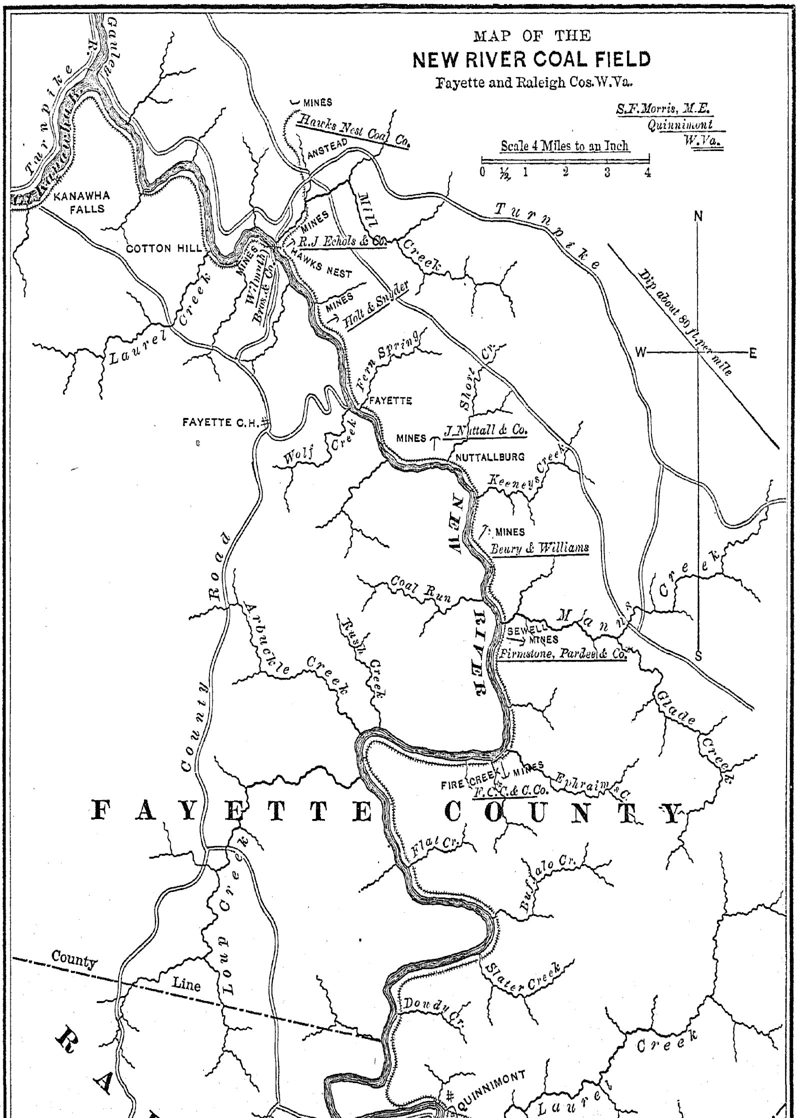
The geological position of these coals is *in* the Conglomerate (No. XII), and the name "Inter-conglomerate" by which they are known was proposed, I believe, by Professor Fontaine, of the University of West Virginia.

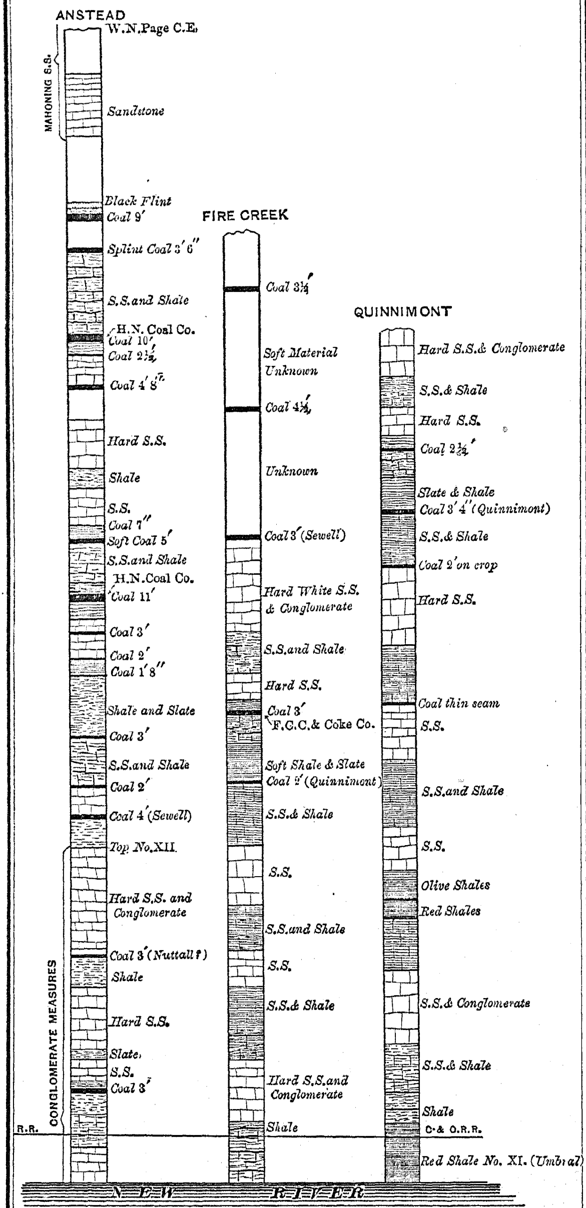
The thickness of the Conglomerate on the New River is not yet certainly known, but from the top of the red shale of XI at Quinnimont to the top of the Conglomerate shown on the Hawk's Nest section is about 1450 feet, of which 1300 feet is to be seen at Quinnimont.

The elevations on all the accompanying sections were obtained by the levelling instrument, with the exception of the two upper coal seams on the Fire Creek section, which were ascertained by an aneroid barometer. For the very complete section at Hawk's Nest and Anstead I am indebted to Mr. W. N. Page, superintendent of the Hawk's Nest Coal Company, who has proved all the coal veins shown on the section, and carefully measured all the intermediate rocks. This section exhibits the "Lower Productive Measures" (from the Conglomerate to the Mahoning sandstone), which are here 1200 feet thick and contain sixteen veins of coal, having an aggregate thickness of 67 feet, and of these sixteen veins, eleven are workable or contain 3 feet of coal and upwards.

The measures have a dip of from 75 to 100 feet per mile to the northwest, and the regularity of dip and strike is of great value in determining the proper location of new mines, in laying out the workings, and providing for permanent and cheap means to secure ventilation and drainage. The coal is soft, is easily and cheaply mined, and no expensive machinery is required for handling the coal, or for ventilation or drainage. The amount of dead work in the mines is small, as the coal veins have excellent roofs, and all the mining expenses can be brought to as low or a lower figure than in other regions.

The coal has a high reputation as a steam coal, and is fully equal to the best-known steam coals in the country, but it is the coking property which makes it very valuable, and which is rapidly bringing it into use in the great iron region of the Ohio Valley, several of the largest and best-known furnaces on the Ohio having used the


MAP OF THE
NEW RIVER COAL FIELD
Fayette and Raleigh Cos. W.Va.
S.F. Morris, M.E.
Quinnimont
W.Va.
Scale 4 Miles to an Inch
0 ½ 1 2 3 4
N
W
E
S
Dip about 80 ft. per mile
Turnpike
Gauley R.
Kanawha R.
KANAWHA FALLS
COTTON HILL
MINES
Hawks Nest Coal Co.
ANSTEAD
R.J. Echols & Co.
HAWKS NEST
Wilmoth Bros. & Co.
Holt & Snyder
Mill Creek
Turnpike
Laurel Creek
Fern Spring
FAYETTE
Short Cr.
FAYETTE C.H.
Wolf Creek
J. Nuttall & Co.
NUTTALLBURG
Keeneys Creek
NEW RIVER
Beury & Williams
Coal Run
Rush Creek
Arbuckle Creek
County Road
Manns Creek
SEWELL
Firmstone, Pardee & Co.
Glade Creek
FIRE CREEK
F.C.C. & C. Co.
Ephraim C.
FAYETTE COUNTY
Flat Cr.
Buffalo Cr.
Loup Creek
Slater Creek
County Line
Dowdy Cr.
RALEIGH COUNTY
QUINNIMONT
Laurel Creek
Pa. & Va. I. & C. Co.
C. & O. R.R.
County Line
Meadow Cr.
Piney Creek
Glade Creek
RALEIGH C.H.



VERTICAL SECTIONS

Scale 300 feet to an Inch

New River coke for nearly two years with the best results. It will "stand up" under a heavy burden in the furnace, it contains a high percentage of carbon with very little ash and sulphur, and extended use and experience has proved it to be one of the best and most economical furnace fuels in use. Mr. J. H. Bramwell, who has used this coke in the Quinnimont Furnace for over five years, says: "In using the Virginia brown hematites, the quantity of fuel required will vary with the percentage of metallic iron, silica, and alumina contained in the ore. From $1\frac{1}{4}$ to $1\frac{1}{2}$ tons of 48-hour coke is a usual average. Using 72-hour coke and an ore with 50 per cent. metallic iron, 5 per cent. silica, and of an aluminous nature, 1 ton and even less has produced a ton of iron at Longdale Furnace."

The hematites occurring in juxtaposition to sand rock, and imbedded in a matrix derived from this source, contain on an average, however, 45 per cent. metallic iron and from 15 to 20 per cent. silica. These require a high percentage of lime to flux them, and fully $1\frac{1}{2}$ tons of 48-hour coke for their reduction. The introduction of Whitwell stoves and more carefully prepared ores will reduce this high consumption of fuel to 1 or $1\frac{1}{4}$ tons of 48-hour coke.

The increasing demand for this coke is developing the region, and new mines are being opened and new ovens are under contract at several places, and in a short time the New River district will rank among the most important coke-producing districts in the country, and will attain this position by reason of the superior quality of its coke, which specially adapts it for metallurgical uses, and the improved transportation facilities, which will enable the coke manufacturers to furnish a regular supply to consumers in the great market of the Ohio Valley.

The coke is made exclusively in beehive ovens, and experience and experiment seem to have proved thus far that coke thus made is superior for furnace use to that made in any other form of oven. It is made in the usual manner by dumping about 3 tons of coal in the oven through a hole in the top, spreading it evenly over the floor, and coking it 48 hours, except on Fridays and Saturdays, when a larger amount of coal is charged, and the coking continued for 72 hours. The coal yields from 63 to 64 per cent. of coke, one manufacturer claiming a yield of over 65 per cent., which is a very high yield for beehive ovens.

Below is a table of analyses of several New River cokes and an analysis of standard Connellsville coke, by which it will be seen that the New River coke contains about 5 per cent. more carbon than the Connellsville, and only one-half as much ash:

	Carbon.	Ash.	Sulphur.	Moisture.	Chemist.
Connellsville	87.26	12.00	0 74		2d Geol. Survey of Pa.
Quinnimont	93.85	5.85	0 30		J. B. Britton.
Quinnimont	93.11	5.94	0.82		Prof. Egleston.
Fire Creek	92.18	6.68	0 61	0.11	Dr. Ricketts.
Sewell	93.00	6.73	0.27		C. E. Dwight.
Nuttallburg	92.22	7.53	0.92		C. E. Dwight.

The largest works in the district are those of the Pennsylvania and Virginia Iron and Coal Company at Quinnimont. These works were begun in 1873, and consist of a blast furnace, 15×60, 100 bee-hive coke ovens, the coal mines, and all necessary shops and buildings, including a small foundry. The furnace was blown in in the latter part of 1874, and has been running ever since, except when blown out for repairs.

The output for the mines this year will be about 60,000 tons of coal, and of the ovens 30,000 tons of coke, that portion of the coke which is not used in the furnace being shipped to Western markets.

The coal mined is from a seam a little over 3 feet thick, with no slate partings, lying high up in the mountain, over 1000 feet above the New River, and is brought down the mountain on an incline, 2100 feet long, to the coke ovens. It is a soft, semi-bituminous coal, makes a very bright, hot fire, leaving but little ash, and is an excellent steam coal. Nearly all the coal mined is used in the coke ovens to supply the furnace with fuel, the coal yielding 63 per cent. of coke.

This coal has been analyzed by Professor Egleston and J. B. Britton, with the following results:

ANALYSIS OF QUINNIMONT COAL.

	No. 1. Britton.	Lump coal, No. 2. Egleston.	Slack, No. 3. Egleston.
Fixed carbon	75.89	79.26	79 40
Volatile matter	18.19	18.65	17.57
Ash	4.68	1.11	1.92
Sulphur	0.30	0.23	0.28
Water	0.94	0.76	0 83

Ten miles down the river below Quinnimont, Mr. J. H. Bramwell has a very good opening on the Quinnimont seam, and has contracted for the construction of coke ovens to supply fuel for a large furnace on the Ohio.

Six miles further west are the works of the Fire Creek Coal and Coke Company, which were opened in 1876, and have been running ever since. This company has an excellent plant of 60 coke ovens, and also ships a considerable amount of coal for steam pur-

poses. This coal is in a higher seam than the Quinnimont, and is not so friable. Its analysis is as follows:

ANALYSIS OF FIRE CREEK COAL.

Fixed carbon,	75.02
Volatile matter,	22 34
Ash,	1 47
Moisture,	0.61
Sulphur,	0.56
	100 00

Two miles beyond Fire Creek, at Sewell Station, are the works of the Longdale Iron Company (Firmstone, Pardee & Co.), who here have a plant of 40 ovens to supply their Longdale Furnace with fuel. Preparatory to the erection of another furnace this company is engaged in building 60 more ovens, and by the first of June will have 100 ovens in operation. They are working a vein of excellent coal, and the coke does remarkably good work in their furnace at Longdale.

An analysis of their coal by C. E. Dwight gave the following result:

ANALYSIS OF LONGDALE COAL.

Fixed carbon,	72.32
Volatile matter,	21.38
Ash,	5 27
Water,	1.03
Sulphur,	0.27
	100.27

Two miles beyond Sewell is the mine of Beury & Williams, who have a very good opening, showing in several places four feet of coal free from slate and of excellent appearance. No analysis has been made of this coal, which is shipped to Eastern markets for steam purposes.

Three miles further on is the Nuttallburg mine of Mr. John Nuttall, of Pennsylvania, who is also an operator in the Clearfield region. Mr. Nuttall opened this mine in 1873, and ships his coal to Eastern markets, and he also has a plant of 50 coke ovens, the coke being shipped down the Ohio River for use in blast furnaces. Two analyses of this coal are given, one by C. E. Dwight, made two or three years ago, and one by Professor Egleston, made recently. The ash in Mr. Dwight's samples is unusually small.

ANALYSIS OF NUTTALLBURG COAL.

	No. 1.—Dwight.	No. 2.—Egleston.
Moisture,	0.34	1.35
Volatile matter,	29.59	25.35
Fixed carbon,	69.00	70.67
Sulphur,	0.78	0.57
Ash,	1.07	2.10
Phosphorus,	——	0.08

About four miles west of Nuttallburg a new mine is being opened by Holt & Snyder on the Nuttall vein, about 300 feet above the level of the railroad. No analysis of their coal has been made, but it probably does not differ materially in its composition from the Nuttallburg. At Hawk's Nest, two miles further west, and thirty miles from Quinnimont, the railroad crosses New River, and the Nuttall vein is worked on both sides of the river, this seam being here only about 75 feet above the railroad, having fallen fully 1500 feet in the thirty miles from Quinnimont. The following analysis was made in 1877 by Professor J. W. Mallet, of the University of Virginia:

ANALYSIS OF HAWK'S NEST COAL.

Moisture,	0.93
Fixed carbon,	75.37
Volatile matter,	21.83
Ash,	1.87
Sulphur (in ash),	0.26

With these mines at Hawk's Nest we have completed the list of the coking-coal mines on the New River, nine in all, the output for 1880 being estimated at 200,000 tons coal, and by midsummer there will probably be 350 coke ovens in operation, with every indication of a rapid growth of this important industry.

At Anstead, about three miles northeast of Hawk's Nest, is the mine of the Hawk's Nest Coal Company, which was opened in 1873 by the Gauley-Kanawha Coal Company. This company ships about 200 tons per day to Eastern markets, where it meets with much favor as a locomotive fuel. Although this mine is in the New River region, the coal is mined from a vein in the Lower Productive Measures, about 400 feet above the Conglomerate, the seam being 11 feet thick, with 2 very thin slate partings. The following analysis of this coal is taken from *Resources of West Virginia*, by Fontaine and Maury:

ANALYSIS OF ANSTEAD COAL.

Carbon,	63.10
Volatile matter,	32.61
Water,	1.40
Ash,	2.15
Sulphur,	0.74
	100.00

A NEW AIR-COMPRESSOR.

BY E. GYBBON SPILSBURY, PHILADELPHIA.

THE introduction of underground machinery in mines, and especially the invention of the rock drill, called attention to the necessity for some motive power to drive them. The use of steam generators underground is as a general thing inadmissible, owing to the difficulties of ventilation and the transportation of fuel, and the great loss of power by condensation in the pipes, when surface generators are used, added to the inconvenience of getting rid of the exhaust steam, has precluded the direct use of steam for this class of machinery. Water as a power has with very few exceptions proved a failure, owing to its weight and incompressibility, and therefore the attention of mining engineers has very naturally turned to the use of compressed air. This agent not only answers all the requirements for power, but has the double advantage of greatly assisting in thoroughly ventilating remote workings, and in cooling the underground atmosphere. It is therefore clear that the successful introduction of underground machinery is dependent on the economical production of good and efficient air-compressing machinery. Although many improvements have of late years been introduced in this class of machines, the most of them are open to serious objections, the chief of which are the very *small duty they perform for the power expended*, and their great expense, due to the complexity of their construction. This latter fault is the chief reason why the introduction of rock drills and other underground engines is so limited, as comparatively few mines can afford to expend the required capital for the installation of a complete compressing plant. But with the introduction of a cheap and effective compressor, there is no doubt that power drills will become as generally used in mines as the hand drills are now.

The great excess of power required in most machines to compress air over that given out again by the use of the air thus compressed (amounting as it does in most cases to sixty per cent.) is well known to all mining engineers. Of course a portion of this loss is due to the friction of the air through the pipes, and will always occur however effective the machine used may be, but the chief cause of the

discrepancy is probably due to the contraction of the compressed air caused by the radiation of heat from the air during its passage through the receiver and the pipes. It is self-evident that power is expended in generating this heat, and this heat again reacting on the compressed air increases its tension, and consequently its resistance to the compressing power. This increase of tension advances in a regular ratio amounting to 0.00204 of the pressure for each degree of heat, on the Fahrenheit scale. Now in the general run of compressors the temperature of the air at its discharge is in the neighborhood of 180° F., and in many cases much higher, supposing the tension of the air to be 80 pounds, and the atmospheric temperature 60° F. The moment the air enters the receiver and pipes, radiation of heat commences, and, as the pipes are generally exposed to the air, progresses very rapidly until an equilibrium is established between the air outside and inside the pipes. This radiation amounts to a loss of 120 degrees of heat, and consequently a loss of tension $= 120 \times 0.00204 \times 80 =$ 19.584 pounds. Thus, although power enough has been expended to raise the tension to 80 pounds, it has only practically raised it to 60.416 pounds. If the air can be used immediately at its discharge from the compressor, this loss of power due to radiation would be but small, but in nearly every case the air has to be carried to great distances before it can be used.

Another general cause of loss of power is due to the friction of the air passing through the valves. From the necessity that these valves should withstand great pressures, it has been found requisite, or at least advisable, to increase their number as much as possible, but at the same time to reduce their area in order to protect them from inordinate wear and tear. Thus the resistance of the air by the contraction of the air-ways has been much increased. This again has been further enhanced by the introduction of weights or springs to assist in the rapid closing of the valves.

One other cause of loss of power in all piston-compressors is due to the fact that in no case can the piston run to the extreme end of the cylinder, and therefore there is always left in the cylinder a small amount of highly compressed air, which on the return stroke expands and so fills up again a portion of the space which should theoretically be occupied by fresh atmospheric air. The higher the pressure required the greater is the loss due to this cause.

The necessity for the large number of valves and working parts, added to the complex arrangements required for the introduction of

a constant supply of cold water around all the working parts of the compressor, to prevent over-heating and to counteract somewhat the loss due to radiation, has greatly increased the cost of this class of machines, and has therefore, as stated above, practically placed them out of the reach of the average miner.

There has, however, lately been introduced in this market a new air compressor known as the "Atlas," which from its novel construction and very great simplicity seems practically to do away with most of the objections above cited. Although comparatively new in this country, this machine, which is the invention of a Mr. William Johnston, has already been introduced largely in England and the continent of Europe and appears everywhere to meet with the most thorough success.

The principle on which it works is the compression of the air against a surface of water, instead of a metal surface, and as this water is being constantly renewed it insures the instantaneous absorption of all the heat envolved during compression. Although this principle of compression is by no means new, being in fact one of the oldest systems ever practically used, still the construction and details of the machine are both novel and ingenious. As can be seen by the accompanying drawings the machine consists of one or more horizontal cylinders, suspended on a stationary arbor or shaft, to the under side of which on its whole length inside the cylinder is attached a diaphragm or piston. Above the shaft is situated a sector-shaped chamber, which also extends the whole length of the cylinder, and is divided into two by a middle partition, and is fitted with the necessary valves. The cylinder oscillates on this fixed shaft.

In the annexed diagrams Figs. 2, 3, 4, and 5, *A*, *A*, *A*, *A*, is the cylinder; *B*, *B* are the cylinder heads; *C* is the shaft or arbor; *D* is the fixed piston or diaphragm; *E* is the cast iron sector-shaped valve-chamber, divided into two separate compartments by the partition *F*; *G* and *G'* are the valves, connecting the inside of the cylinder with the outside atmosphere by the opening, *e''*, through the valve-chamber *E*; *f*, and *f'*, valves connecting the interior of the cylinder by the valve-chamber, *f''*, with the air-receiver; *b'*, crank-pin, to which is attached the connecting-rod of the driving-engine; *N*, *N*, hydraulic packing, taking the place of stuffing-boxes.

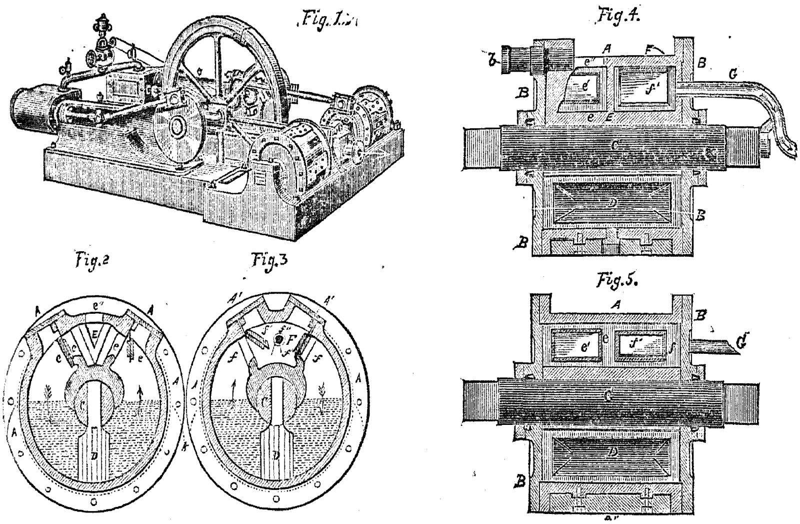
The action of the machine is as follows:

The cylinder being half filled with water, begins to oscillate in the direction of the arrow (Fig. 3). The valve *f* closes, and the air con-

tained in the space above the face of the water is forced to enter the chamber F, through the valve f'. At the end of the stroke the valve f lies on the water surface, and the whole of the air with any surplus water has entered the chamber F, and passed on into the receiver.

In the meanwhile the atmospheric air has rushed in through the opening e'' and the valve e', and has filled the space on the other side of the cylinder. On the return stroke the reverse motion takes place.

From inspection of the drawings it will be seen that the con-



struction of these machines is extremely simple. They require no fine work and but very little fitting. The plunger or diaphragm does not fit tight to the cylinder, and the water having free play past it, and also being forced around the shaft and under the bearing of the valve-chamber, tends to act as a perfect lubricator, and also actually lifts the cylinder from the fixed shaft. In this manner the amount of friction is reduced to a minimum. The valves are extremely simple in construction, being nothing more than ordinary rubber clock or door valves, and as they have a very large area the friction of the air in its passage through them is very much lessened. A small stream of water is fed in constantly with the air, and as the heat of the compressed air is instantaneously absorbed by this water, the air passes out from the machine at a temperature very little above

that of the water introduced, and in many cases considerably below that of the atmosphere.

The above-described single-cylinder machine is adapted to all purposes up to a pressure of say 70 pounds to the square inch. Higher pressures are found to wear out very rapidly the valve facings, and therefore for the cases where heavy pressures are required a double-cylinder machine as shown in Figs. 4 and 5 is used. The internal construction of both these cylinders is exactly the same, and they oscillate together on the same shaft. The second cylinder, *a, a*, is made proportionally so much smaller than the first one that its compartments will just be filled by the volume of air discharged, under a given pressure from the large cylinder. Thus if we say we will not expose the valves of the large cylinder to a pressure exceeding four atmospheres, or 58.80 pounds, we should make the capacity of the small cylinder one-fourth that of the large one. Then if the ultimate tension required in the receiver is, say 100 pounds per square inch, it is evident that the only pressure the valves, e, e', will have to stand will be $100 - 58.80 = 41.20$ pounds. By this compound system enormous pressures can be obtained, since the gradual reduction in the size of the valves, as the volume of the air decreases, greatly increases their power of resistance.

From their simplicity these machines ought to be supplied to the market at prices far below those of any other pattern; and they have another advantage, that there is no working part likely to get out of order which cannot be at once replaced by any ordinary mine-blacksmith.

PHILADELPHIA, September 11th, 1879.

NOTE ON THE DEFREEST JOURNAL-BEARING.

BY J. C. PLATT, JR., WATERFORD, NEW YORK.

I DESIRE to call the attention of the Institute to what is believed to be a new improvement in the construction of journal-bearings, having for its object the reduction of original cost as well as cost of renewals.

In an ordinary railroad car we find the space between the top of the axle and the top of the box is filled with a "brass," weighing from seven to ten pounds. Of this amount of metal a comparatively small portion is subjected to the wear which it is intended to meet. The balance is, practically, a filling material, which from habit has long been constructed of the same metal.

The Defreest bearing is made of three parts, namely, a malleable-iron back, a bronze bearing-piece, and between them a filling-piece of sheet lead. The bronze takes the wear of the axle, the iron fills up the space and acts as a backing to the bronze, and the lead, being soft, adapts itself to the inequalities in the surfaces of the iron and bronze castings, thus insuring a uniform bearing on all points, and making it unnecessary to bore out the iron or turn off the bronze to bring them to a perfect contact at all points. These three parts are joined together by two soft brass rivets, with counter-sunk heads, to prevent separation in handling. A groove is cast in the iron, and a corresponding rib on the bronze, to prevent the latter from turning with the axle when in place.

In renewals it is ordinarily necessary simply to drive out the rivets and renew the bronze and lead. The iron part is not subjected to much wear and should last for years.

It is evident that the first cost of these journals, and also the cost of renewals, must be much less than the old form. This difference amounts, at the present price of materials, to $9.60 for an eight-wheel car, in first cost, and $4.72 for renewals.

These bearings have been in use, and found to do what is claimed for them.

PROCEEDINGS

OF THE

ANNUAL MEETING IN NEW YORK.

FEBRUARY, 1880.

THE opening session was held on Tuesday evening, February 17th, in the house of the American Society of Civil Engineers.

The President of the Institute, Mr. E. B. Coxe, after a few introductory remarks, congratulating the members on the general prosperity in all the branches of the professions and manufactures represented by the Institute, called on Mr. E. F. Loiseau, of Philadelphia, for the first paper, On the Successful Manufacture of Compressed Fuel at Port Richmond, Philadelphia.

Professor Prime called attention to the fact that the work of the United States Testing Board had been left unfinished at a point when valuable results had just been reached, and offered the following resolutions:

WHEREAS, The American Institute of Mining Engineers has seen with regret that the work of the United States Testing Board has been suspended through the lack of an appropriation by Congress, which appropriation does not seem likely to be renewed, and

WHEREAS, It seems most important to the vital interests of our iron and steel manufactures that the experiments commenced by the United States Testing Board should be carried to successful completion;

Resolved, That a committee of three be appointed, to act with a like committee to be appointed by the American Society of Civil Engineers, to carry out to completion the work commenced by the United States Testing Board.

Resolved, That the American Society of Civil Engineers be requested to co-operate with this society by the appointment of a like committee.

Resolved, That the committee have authority to request, in the name of the American Institute of Mining Engineers, contributions to a fund to be devoted to the completion of the experiments aforesaid.

Resolved, That the Council of this Institute be authorized, whenever requested by the committee so appointed, to petition Congress for the free use of the testing machine at the United States Arsenal at Watertown.

Resolved, That a committee be appointed by the President, to select the aforesaid committee of experts.

Resolved, That the President of the Institute have authority to fill any vacancies which may occur in the committee.

The resolutions were unanimously adopted, and President Coxe appointed the following committee in accordance with the resolutions: Messrs. A. L. Holley, James Park, Jr., and J. F. Holloway. The President also appointed Messrs. Heinrich, Mickley, and Spils-

bury scrutineers to examine the ballots for officers and report at a subsequent session.

Professor R. H. Richards then read a paper entitled Notes on Battery and Copper-plate Amalgamation, from the Mining Laboratory of the Massachusetts Institute of Technology.

Before the adjournment of this session there were distributed to the members present an illustrated programme of the meeting prepared by the Local Committee of Arrangements, and cards of invitation to the reception of Mr. and Mrs. J. A. Burden and the Bullion Club.

The second session was held on Wednesday morning.

The Secretary read a paper by Professor P. H. Mell, Jr., of Auburn, Ala., on The Claiborne Group and its Remarkable Fossils. This paper was accompanied by a large collection of the fossils, which the author desired to be given to the committee having in charge the collections designed for the German Government.

The following papers were also read at this session:

Blast Furnace Working, by Julian Kennedy, of Braddock, Pa.

Notes on the Blast Furnace, by J. M. Hartman, of Philadelphia.

The Puddling Process, Past and Present, by Percival Roberts, Jr., of Philadelphia.

Dr. A. L. Holley showed a small specimen of iron found under the Egyptian Obelisk, which is now on its way to this country. It had been analyzed by Dr. Wendel, of Troy, with the following results:

Iron,	98.738
Carbon,	0.521
Sulphur,	0.009
Silicon,	0.017
Phosphorus,	0.048
Manganese,	0.116
Nickel and cobalt,	0.079
Copper,	0.102
Calcium,	0.218
Magnesium,	0.028
Aluminium,	0.070
Slag,	0.150
Total,	100.096

The specimen was too small for physical tests. A clean fracture showed a rather highly carburized and granular, but tough-looking metal.

Mr. R. B. Harrison, Superintendent of the Assay Office of Helena, Montana, exhibited a large collection of gold crystals from the

gulches of Montana, the property of Mr. T. H. Kleinschmidt, of Helena; also a collection of large and small nuggets. Mr. Harrison gave a brief description of the occurrence and production of gold in Montana.

Dr. Henry Wurtz, of Hoboken, exhibited and described specimens of Huntilite and Macfarlanite from Silver Islet.

President E. B. Coxe read a communication on the use of carbonate of soda for the prevention of boiler scale.

The third session was held on Wednesday afternoon. The Secretary read the report of the Council, as follows:

"The Council of the Institute makes the following annual report of the work of the past year.

"The receipts and disbursements as shown by the accounts of the Secretary and Treasurer, duly audited, were as follows:

"*Secretary's and Treasurer's Statement of Receipts and Disbursements from February* 14*th*, 1879, *to January* 31*st*, 1880.

Dr.		
Balance at last statement,	$124 77	
Received for dues from members and associates,	5942 00	
" " life membership,	100 00	
" from sale of publications,	598 99	
" for author's pamphlets,	203 24	
" for binding Transactions,	139 05	
Interest,	14 60	
		$7122 65
Cr.		
Paid balance due for printing Vol. VI, Transactions,	$627 91	
Paid for printing Vol. VII, Transactions,	1362 66	
" " pamphlets,	768 80	
" " circulars, etc.,	133 40	
" engraving,	445 90	
" binding Transactions,	123 00	
" stationery,	68 48	
" postage,	635 03	
" telegrams,	6 20	
" rent of hall (Baltimore meeting),	38 00	
" freight and expressage,	110 49	
" exchange of fire-proof safe,	10 00	
" furniture,	19 52	
" insurance,	37 50	
" Secretary's salary,	2000 00	
" " expenses attending meetings,	101 55	
		6488 44
Excess of receipts over expenditures,		$634 21
Deduct for dues received in advance,		620 00
Surplus for the year ending January 31st, 1880,		$14 21

"Thus the expenses of the year have not only been met by the receipts, but the deficit of $500 from the previous year has, in addition, been paid off. The Institute is to be congratulated on the favorable showing of its finances.

"Three meetings have, as usual, been held, at Baltimore, Pittsburgh, and Montreal, and have been attended with the usual enthusiasm and profit. At these meetings there were elected 133 members and 19 associates. During the year 13 members and associates have resigned, one, Mr. J. T. Fuller, of Wilkes-Barre, has died, and 58 have been dropped from the lists, after due notification, for non-payment of dues. Our membership now comprises 5 honorary members, 53 foreign members, 616 members, and 118 associates; total, 792.

"The number of professional communications presented during the year was 56. These do not include the discussion of Dr. Dudley's papers on Steel Rails, at the Baltimore and Pittsburgh meetings, in which 15 members participated. These papers, with the discussions, have been separately printed in pamphlet form, and widely distributed to those interested at home and abroad.

"The greater part of the papers presented have been issued in pamphlet form, *subject to revision*, and have appeared, or are yet to appear in the annual volume of Transactions. Volume VII of the Transactions, covering the period from May, 1878, to February, 1879, was issued last December and was distributed as usual.

"The *Employment Agency* of the Institute, which the Secretary was authorized to organize at the Montreal meeting, bids fair to be of value in finding professional employment for members of the Institute, and of great convenience to those who wish to employ mining engineers, metallurgists, geologists, or chemists. It is too soon, however, to comment on the working of the scheme.

"At the last annual meeting, Mr. J. S. Alexander, Chairman of the Museum Committee of the Institute, made a final report of the condition of the Museum, and the Council is now happy to announce that the arrangements contemplated at that time, involving the transfer of the collections to the Pennsylvania Museum and School of Industrial Art have been consummated. The formal transfer was publicly made at Memorial Hall, Philadelphia, on March 26th, 1879. Since that time the installation of the collection has been completed, as far as possible, by the officers of the Pennsylvania Museum, aided by Mr. C. A. Young, the member of the Institute appointed by the terms of the agreement to serve on the Committee

of the Museum, and the exhibit is now a creditable one both to the Institute and the Pennsylvania Museum.

"In this connection it may be said that the request made by Mr. Alexander for contributions of Claiborne (Ala.) fossils for presentation to the German Government, has been handsomely responded to by one of our members, Professor P. H. Mell, Jr., of Auburn, Ala., who has sent a large and valuable collection for this purpose.

"In the destruction of Pardee Hall of Lafayette College last June, the Institute lost its entire library, consisting for the most part of the proceedings of scientific societies and technical journals, acquired in exchange for its Transactions. This loss the Council hopes will be mainly temporary, for many of the sets thus lost have been kindly replaced and more are promised. At the next annual meeting a full report of the condition of the library will be given.

"In conclusion the Council is happy to be able to record the steady growth of the Institute in all the elements of true prosperity and usefulness."

The amendments to the rules, of which notice was given at the previous meeting, to abolish the distinction between home and foreign members, were adopted unanimously.

The changes involved in the rules are as follows: Omit in Rule I the phrase "and members and associates permanently residing in foreign countries;" in Rule II, the phrase "and members and associates permanently residing in foreign countries, excepting Canada, shall be liable to such annual and other payments only as the Council may impose to cover the cost of supplying them with publications;" and in Rule V, the phrase "or foreign members or associates." It was announced on behalf of the Council that these changes would not alter the relations of the present foreign membership to the Institute, but that in the future no more foreign members, so-called, would be elected.

The following members and associates were unanimously elected:*

E. C. Appleton,	Allentown, Pa.
Charles A. Bauer,	Springfield, Ohio.
Jackson Bailey,	New York City.
Theo. A. Blake,	New Haven, Conn.
Alfred P. Boller,	New York City.
J. B. Brinsmade,	New York City.
Henri M. Braëm,	New York City.
Harvey B. Chess,	Pittsburgh, Pa.
Richard E. Chism,	Norristown, Pa.

* In this list are included those elected at the final session.

James R. Cooper,	Houghton, L. S,. Mich.
A. W. Crookston,	Glasgow, Scotland.
L. L. Crounse,	Kingston, N. Y.
Gram Curtis,	New York City.
W. B. Devereux,	King's Mountain, N. C.
John Duncan,	Calumet, L. S., Mich.
Charles E. Emery,	New York City.
H. H. Fisher,	Allentown, Pa.
George G. Francis,	Montreal, Canada.
James Gayley,	Catasauqua, Pa.
Robert Grimshaw,	Philadelphia, Pa.
John H. Grove,	Danville, Pa.
C. H. Hall,	Ishpeming, Mich.
Henry J. Hall,	New York City.
J. F. Hartranft,	Philadelphia, Pa.
E. A. Hitchcock,	St. Louis, Mo.
Thomas Hoatson,	Calumet, L. S., Mich.
W. A. Hooker,	New York City.
Fred. F. Hunt,	Capelton, Canada
J. E. Johnson,	Longdale, Va.
John T. Jones,	Sharon, Pa.
Clarence King,	Newport, R. I.
James G. Knap,	Philadelphia, Pa.
John H. Knox,	Andover, N. J.
James C. Lewis,	Portsmouth, Ohio.
John C. Lewis,	St Louis, Mo.
Edwin Ludlow,	Philadelphia, Pa.
Arthur M. McComb,	Philadelphia, Pa.
Ed. A. McCulloh,	Aurora, Ill.
C. F. Manness,	Scranton, Pa.
G. W. Maxson,	Auburn, Alabama.
C. C. Morgan,	Denver, Col.
Jay C. Morse,	Marquette, Mich.
William D. Mullin,	Pittsburgh, Pa.
Charles M. Noble,	Anniston, Ala.
C. C. Newton,	Cleveland, Ohio.
Wm. N. Page,	Ansted, W. Va.
Charles T. Porter,	Newark, N. J.
R. D. Rickard,	Middletown, N.Y.
William Barret Ridgely,	Springfield, Ill.
Eben E. Robinson,	St. Albans, Vt.
W. T. Sprague,	Scranton, Pa.
David Townsend,	Philadelphia, Pa.
Charles R. Westbrook,	Ogdensburgh, N.Y.
Chas. B. Wheelock,	Santa Fé, New Mexico.
Samuel Whinery,	Wheeler Station, Ala.
David Williams,	New York City.
W. C. Witherbee,	New York City.

ASSOCIATES.

F. E. Bachman,	Lafayette College, Easton, Pa

Chas. P. Bleecker,	New York City.
Alex. B. Coxe,	Drifton, Jeddo P. O., Pa.
Austin Farrell,	Lafayette College, Easton, Pa.
Frank Klepetko,	New York City.
W. B. Kunhardt,	New York City.
A. E. Swain,	Cleveland, Ohio.

The status of the following associates was changed to member: Walton Ferguson, F. A. Genth, Jr., Benjamin Marshall, and T. S. Mathis.

Dr. C. William Siemens, F.R.S., was duly proposed for honorary membership in consideration of his distinguished services in the metallurgy of iron and steel. This proposal was approved by the Council, and Dr. Siemens was unanimously elected an honorary member of the Institute.

The scrutineers appointed at the first session to examine the ballots for officers reported the following elected:

PRESIDENT.

William P. Shinn,	St. Louis, Mo.

VICE-PRESIDENTS.

James A. Burden,	Troy, N. Y.
Charles B. Dudley,	Altoona, Pa.
Persifor Frazer, Jr.,	Philadelphia, Pa.

MANAGERS.

James C. Bayles,	New York City.
W. S. Keyes,	San Francisco, Cal.
Percival Roberts, Jr.,	Philadelphia, Pa.

TREASURER.

Theodore D. Rand,	Philadelphia, Pa.

SECRETARY.

Thomas M. Drown,	Easton, Pa.

The following papers were then read:

Fuel-gas and the Strong Water-gas System, by Dr. Henry Wurtz, of New York.

The Coal and Iron Fields of North Staffordshire, England, by W. Hamilton Merritt, of St. Catharines, Canada.

Invitations were received from Mr. P. H. Dudley for the members to visit his dynagraph car, at the Grand Central Depot, and from Mr. N. S. Keith to visit the works of the Fuller Electrical Company in Brooklyn.

On Wednesday evening Mr. and Mrs. James A. Burden received the members of the Institute and their ladies and friends at their house on Fifth Avenue.

The fourth and concluding session was held in the metallurgical lecture room of the School of Mines of Columbia College, on Thursday morning.

The following papers were read:

Notes on the Siemens Direct Process, by Dr. A. L. Holley, of New York.

The Water-gas Furnace of W. A. Goodyear, by Professor B. Silliman, of New Haven, Conn.

The Mineral Resources of Southwestern Virginia, by C. R. Boyd, of Wytheville, Va.

The Electric Light as Applied to Mining, by N. S. Keith, of New York.

Mr. G. W. Maynard exhibited some interesting gold specimens from California, which were commented on at length by Professors Egleston, Silliman, Hunt, and Kerr.

Mr. C. Constable exhibited and described specimens of overblown iron, accompanied by fragments of the basic lining, from the Bessemer works of Ruhrort, Germany.

Professor Egleston, chairman of the committee appointed to examine the condition of the collections of the Institute, which had been recently transferred to the Pennsylvania Museum and School of Industrial Art in Philadelphia, made the following report of the condition of the collections:

"A majority of the members of the Committee on the Museum, appointed at the Pittsburgh meeting, met in Philadelphia, at Memorial Hall, and examined very carefully the collections of the Institute. Those which require to be covered are in cases, and all of them are well cared for. The Kirkaldy collection is in a perfect state of preservation, and is as well displayed as it was during the Centennial Exhibition. The Stassfurth collection of salts has undergone slight damage from the breaking of some of the glass bottles and the cracking of others, owing to the decomposition of the contents from the admission of air. The rocks and minerals are in table cases, but the labels are, as yet, under the specimens. There has not been time to properly arrange and label this collection for the inspection of the public. The duplicates, which consist mainly of some of the Russian irons, and of a very large quantity of the carbonate of iron from

Siegen, are stored in the cellar, and are of no use unless the custom-house restrictions can be removed from them.

"Your committee are happy to say that they think the Trustees of the Pennsylvania Museum and School of Industrial Art are taking all reasonable precautions for the care and preservation of the collections.

"It is greatly to be desired that, as the collections are mostly of use to Philadelphia, the Philadelphia members of the Institute should endeavor to have them properly labelled for public use. If each specimen was labelled in such a way that it could be studied without opening the cases, it would add very largely to their value."

The Secretary announced that the Council had under consideration the holding of a summer meeting at Lake Superior. The committee having the matter in charge would make an announcement of progress as soon as the matter was definitely settled.

The Secretary then read the following papers by title:

The Theory of Rupture by Torsion, and the Determination of Constants for the Formulæ, by Professor R. H. Thurston, of Hoboken, N. J.

The Iron Ore Deposits of the James River, Va., by E. Gybbon Spilsbury, of Philadelphia.

Supplement I, to a Catalogue of Official Reports upon Geological Surveys of the United States and Territories, and of British North America, by Professor Frederick Prime, Jr., of Philadelphia.

Mode of Formation of the Gold Deposits of North Carolina, and the Formation of Mica Veins, by Professor W. C. Kerr, of Raleigh, N. C.

The Mineral Resources of Wisconsin, by Professor Roland D. Irving, of Madison, Wisconsin.

A Note upon the Heat of the Comstock Mines, by Professor J. A. Church, of New York.

The 80-ton Hammer at Creusot, by J. A. Herrick, of Pittsburgh, Pa.

Determination of Sulphur in Sulphides and Coal, and Determination of Silicon in Pig Iron, by Dr. Thomas M. Drown, of Easton, Pa.

Professor Silliman spoke of the pleasure and success of this meeting of the Institute, and offered a resolution, which was unanimously adopted, directing the Secretary to express the hearty thanks of the Institute to Mr. and Mrs. James A. Burden, the American Society of Civil Engineers, the Bullion Club, and the Trustees and Facul-

ties of Columbia College and the Stevens Institute of Technology, for their cordial reception and entertainment; and also to the Local Committee for their thoughtful and happy arrangements for the meeting.

On Wednesday evening the members of the Institute were invited by the Bullion Club to their rooms, 19 West Twenty-fourth Street.

On Friday morning the members assembled at the Stevens Institute of Technology at Hoboken. They were here received by President Morton and the Faculty and taken through the workshops, collections, and laboratories of the institution, and the system of education and training exhibited. A lunch kindly provided refreshed the members before leaving.

A subscription dinner of the members of the Institute, to which a number of guests were invited, was held at the Hoffman House, on Friday evening, and concluded the programme of the meeting.

PAPERS

OF THE

ANNUAL MEETING.

FEBRUARY, 1880.

FUEL-GAS, AND THE STRONG WATER-GAS SYSTEM.

BY DR. HENRY WURTZ, NEW YORK CITY.

HERACLITUS, a sage of antiquity, called the dark philosopher, who refused a throne, preferring a hermit's cell, propounded, twenty-four centuries since, the maxim:

Πολεμος πατηρ παντων.
War (or strife) engenders all things.

This, though probably intended by Heraclitus to apply especially to the internal forces of nature, is often said, with equal reason, of the affairs of men. Controversial strife, whether fortunately or unfortunately, is a crucible through which all new discoveries in science, and all technical applications of science must pass—a test which they must all endure before they can become so vitalized as to germinate, so to speak, take root in the human mind, grow up, and overspread the earth. The greater the number and the power of the elements arrayed against such a growth, and of the influences hostile thereto, the greater should be the inherent vitality of the germ, the more strenuous, skilful, and persistent its cultivators and upholders.

During the decade last past we have had, in spite of the severe stringency of the times, an active growth of this kind in progress, whose prospective importance it would now be difficult to overrate. This is the movement which has for its motive the idea that, generally speaking, *fuel should be gaseous in form*, and which has for its goal the introduction into general public use of gaseous products, made by cheap and rapid processes and on a gigantic scale, distributed throughout our cities and towns in distribution-systems which shall be proportionately gigantic, and sold at prices which will bring such fuel within the means of the poorest householders.

Personally, for ten years past, the writer has never failed, on occasions public or private, to urge his belief that the realization of this idea, deemed by him a certainty of the future, will bring about

important revolutions in human affairs. As once publicly stated, he looks upon it as "the next great stride in civilization," destined to rank at least with the introduction of steam power, railway transportation, the Bessemer process, the electric telegraph, the articulating telephone, and the like events.

The time the writer has long looked for has now at length come, when practical men and moneyed men are working together and organizing, on the basis of the production of gaseous products adapted or adaptable for *fuel*, without direct reference to the use thereof, in a merely vehicular way, as media of convection for illuminant hydrocarbons; this latter being regarded as only subordinate and not essential to the grand aim in view. This, of course, brings into prominence any improved plan that may be found to exist, of generating such gases cheaply and rapidly; and hence what is known as the "Strong Process" at once claimed and has received great and deserved attention.

It is, probably, not necessary that this process should now be described in detail, as Professor Silliman, at the Montreal meeting, explained it. Gentlemen desiring details will find them complete in a printed pamphlet, obtainable from M. H. Strong, Esq., 56 Coal and Iron Exchange, Cortlandt Street, New York city. The writer is now, for the first time, occupied in investigating, chemically, the operation and the products of the Strong apparatus; but this undertaking is yet too recent to have furnished many complete results. The present statement is therefore to be looked on as preliminary only. Experiments to determine directly the *practical* thermic effectiveness of the Strong gas—which is, for most persons, the point of most immediate interest—have not yet been made, though it is possible that some of them may be so in time to be printed with this paper. Results are here given, however, of careful analyses, together with determinations of density, of a sample of Strong gas; the two sets of figures agreeing with each other, as is essential to reliability. The data are thus at length at our command for accurate *theoretical* computations of the thermic energy, or energy of combustion of this gas.

The sample of gas examined is one now contained in a holder of 10,000 cubic feet capacity at Mount Vernon, Westchester County, N. Y. It was made some six weeks since, and has, therefore, stood for this period over water, though apparently without appreciable change in composition. The materials used were egg coal, one third, and waste anthracite screenings, two thirds. Two good analyses of this gas gave, for 100 volumes:

TABLE I.

	No. 1.	No. 2.	Mean.	Density-computation.
Hydrogen	44.55	45.05	44.80	× .0006930 = .0310464
Carbonic oxide	40.29	39 79	40.04	× .0096740 = .3873470
Marsh gas	4.76	4.85	4.80	× .0055300 = .0265440
Carbonic acid	1.11	1.21	1.16	× .0152000 = .0176343
Nitrogen	9.10	9.18	9.14	× .0097134 = .0887805
Oxygen	.19	.09	.14	× .0110560 = .0015478
	100.00	100.17	100 08	

Computed density at 32° F. = .5529000
Four determinations of density by effusion at 32° F., gave a mean = .5512

Traces of sulphuretted hydrogen, doubtless present in this gas when first made, have been removed by the water. There is proof that the holder has otherwise preserved the gas well in the small amount of oxygen present. No diffusion outward could occur, without inward diffusion of air, carrying oxygen.

The above constitutes what may be regarded as a *verified* gas-analysis, agreeing with the experimental density.

In the next table will be found the percentage composition by weight as well as by volume; also the thermic value in Centigrade and Fahrenheit units, or degrees to which one pound (7000 grains) of water may be theoretically heated by one pound of gas.

TABLE II.

	By volume.	By weight.	Computation of thermic value per pound.
Hydrogen	44.80	5.62	× 344.62° = 1935°
Carbonic oxide	40.04	70 06	× 24.03° = 1683.5°
Marsh gas	4 80	4 80	× 130.63° = 627°
Carbonic acid	1.16	3.19	
Nitrogen	9.14	16.06	Centigrade—4245.5°
Oxygen	.14	.28	Fahrenheit—7642°
	100.08	100.00	

One of the striking results of this analysis is the extremely large amount of nitrogen shown. This could only have come from air introduced in the process of manufacture, by reason of imperfection in the experimental apparatus used. This apparatus is so small that the duration of each heat or successive run is necessarily very short—only ten or twelve minutes, instead of thirty or more, as in a working apparatus. The contents of the generator, in products of combustion with air, after each blowing-up with the latter, are swept on to the holder, together with the gases or products of combustion with steam. The former bear, therefore, to the latter a considerable

proportion, appearing to multiply the nitrogen threefold above its proper proportion.

It is the writer's expectation that a perfected fuel-gas production—such as will soon be brought about, now that this manufacture is to be prosecuted on a large scale—will give us a gas containing uniformly less than 3 per cent. of nitrogen by volume, or from 5 to 6 per cent. by weight. The nitrogen in the anthracite yields at most ½ per cent., while that in the steam is inappreciable. Such a gas as this, made with a perfected Strong generator, will have, as shown by the above analyses,—taking into account that 6 per cent. of nitrogen implies an ingress of 7.5 per cent. of air, or 1.5 per cent. of oxygen, which has, therefore, given us 3 per cent. of the carbonic oxide present,—the following composition :

TABLE III.

Computed composition of crude Strong gas, from large working generators.		Reduced to 100 volumes.	Density-computation.	Computed to 100 parts. by weight.	Thermic value.	
					Centigrade.	Fahrenheit.
Hydrogen	45.0	49.6	.0344	6.81	2,347°	4,225°
Carbonic oxide	37.0	40.8	.3947	78.11	1,877°	3,379°
Marsh gas	5.0	5.5	.0304	6.02	786.5°	1,415°
Carbonic acid	1.0	1.1	.0167	3.30	....	....
Nitrogen	2.7	3.0	.0291	5.76	....	....
	90.7	100.0	D. = .5053	100.00	5,010.5°	*9,019°

Or, in round numbers, such gas will have, at 32° F., half the density of air, with a total thermic power of 5000° C., or 9000° F., per pound. (At 60° F., the density will be only .4482.) This standard should be obtained in fair practice; and, with good apparatus in good order, may be reasonably and uniformly expected for *crude Strong fuel-gas,* made from two-thirds screenings, and one-third egg and *unpurified.*

A document referred to below contains evidence that even the imperfect Mount Vernon apparatus has produced gas containing far less nitrogen than the sample analyzed by the writer. This is an analysis by the learned chemist, Dr. P. H. Van der Weyde, as follows:

Hydrogen,	52.3
Carbonic oxide,	39.4
Marsh gas,	4.3
Carbonic acid, / Nitrogen,	} 4.0
Sulphuretted hydrogen,	undetermined.
	100.0

The carbonic acid and nitrogen are here summed up together; but if the carbonic acid be assumed as found in the analyses of the writer the nitrogen becomes 2.9 per cent. only by volume.

The next tabulation represents the product as it will be after purification with lime to remove the 3.3 per cent. by weight of carbonic acid.

TABLE IV.

	By volume.	Density-computation.	Composition by weight.	Thermic value (Fahrenheit) per pound.
Hydrogen..........	50.15	.03475	7.04	4367°
Carbonic oxide...	41.25	.39905	80.76	3493°
Marsh gas	5.56	.03075	6.22	1462°
Nitrogen............	3.04	.02953	5.98	....
	100.00	D. at 32° F. = .494 D. at 60° F. = .437	100.00	9322°

As less than 2 per cent. of the nitrogen out of the six per cent. by weight comes from the anthracite, a full economic view of this product requires further that, as 4 per cent. of the nitrogen costs nothing, it should also be deducted. Should it be found possible therefore, to exclude air wholly, the thermic value of the resulting fuel-gas would be $9322° + 4 \times \frac{9322°}{100-6} = 9719°$.

So perfect a result as this is not, however, at present, counted on.

COST OF PRODUCTION OF STRONG FUEL-GAS.

Pending the experimental investigations on the thermic value of the Strong gas, which the writer has projected, and is now arranging to make, it may be of interest to present some points derived from an important document (not previously before the public), which contains results of experiments upon the amount and cost of production of gas from the experimental Strong apparatus at Mount Vernon, by highly competent gentlemen entirely disinterested in every way. These gentlemen were Charles A. Stanley, Esq., Assistant Superintendent of the Brooklyn City Gas Works, and Professor William D. Marks, of Philadelphia.

The report referred to was made by them August 18th, 1877, to the Brooklyn City Gaslight Company. A copy of this, evidently a *fac-simile* made by impression, has come into the writer's possession. It is in this document that was found the valuable analysis, cited above, of Dr. Van der Weyde. There is copied, also, in this report, a series of experiments previously made by an agent of, and for,

Walter E. Lawton, Esq., of No. 12 Cliff Street, New York, of which latter experiments Messrs. Stanley and Marks remark that they do not give as good an average as their own. Mr. Lawton has since, it is understood, become interested as a promoter of the fuel-gas movement.

In each of these two series of experiments, consisting of a succession of ten-minute runs, the yield of gas ran down gradually. Stanley and Marks obtained at first 1647 cubic feet gas from 63 pounds anthracite, and 1627 cubic feet gas from 63 pounds anthracite. In the Lawton series were obtained 1718 feet from 60 pounds, and 1554 feet from 60 pounds; the mean of these four being 1000 feet from 37.5 pounds; while the tenth runs respectively gave Stanley and Marks 1050 feet from 45 pounds; and Lawton, 1042 feet from 45 pounds; the mean of the last two being 1000 feet from 43 pounds.

Messrs. Stanley and Marks state, however, that "the generator and flues are so small, and the doors so arranged, that the apparatus admittedly cannot run without choking from clinkers." Also: "The apparatus, being the first of its kind, is not so conveniently designed as it might have been; much trouble with clinkering of the fire might be avoided by a design which would admit of stirring the fire." Other imperfections, obvious to these skilled engineers, and readily remediable, are alluded to.

The writer feels perfectly justified, through his past experience in cases of this sort, which has been exceptionally extensive, in estimating the yield obtainable in a well-constructed working apparatus (such, for example, as is now erecting at Yonkers) from the *best* work actually accomplished with this imperfectly-constructed experimental plant; which is, as above, 1718 feet from 60 pounds, or about 1000 feet from 35 pounds coal. For safety, however, let us rather adopt the *four best runs,* two of each set; giving 37.5 pounds per thousand as the yield that may be expected to be fully and continuously realized on a large scale from a perfected plant. The coal used by Stanley and Marks was about one-third egg (used in the generator), worth at that date $5 per ton, and two-thirds of a mixture of dust and pea (in the hopper), worth then $1 per ton. Strong prefers, for obvious reasons, that no pea coal should be used, but all dust or fine screenings, in the hopper; this two-thirds being, or rather including, that portion of the carbon which mainly reacts with the steam, and from which the gas therefore mainly proceeds. Such screenings—an unlimited supply of which, for a century, is

procurable for the mere cost of transportation—may be rated at $1 per ton at most, while egg coal is now about $4.25, though to avoid cavil we will retain the valuation of $5. These data give, for 37.5 pounds anthracite per 1000 cubic feet of fuel gas:

$$\frac{2240 \times 37.5}{\frac{500}{3}+2\frac{100}{3}} = 3.605 \text{ cents; say } \textit{three and two-thirds cents.}$$

The minimum estimates of the Society of Gaslighting, discussed below, put this item at 75 pounds of anthracite at $4.50 per ton, about 15 cents per 1000 feet, which is 400 per cent. above the actual expense shown in the Mount Vernon generator, with a clean fire.

It is to be understood that the 37.5 pounds of anthracite includes *all* coal used for steam-making, and all other purposes in the Mount Vernon apparatus when working fairly. This is expressly set forth by Stanley and Marks; whose allowance, however, for coal consumption, being deduced from the *average* working of the partially clogged generator, during the whole succession of runs, sums up *six cents* per 1000 feet for coal (egg rated at $5). As to labor, in operating the experimental plant, Stanley and Marks state that an engineer at $2.50 per day, a stoker at $1.75, and a helper at $1.25, were occupied four hours and thirty-four minutes in making 13,035 feet of gas; hence they make for labor 17.5 cents per thousand; allowing, however, that "there can be no doubt that, if the process is worked on a large scale, the labor cost can be reduced much below this."

On this point the writer learns from James S. Pierson, Esq., the engineer engaged in constructing the new Strong Gas Works at Yonkers, that he expects these same three men to run at least four working generators, making 200,000 feet each per day of 10 hours, in all 800,000 feet, which will bring down the cost of labor per thousand to less than *two-thirds of a cent.* It is preferred to multiply this for safety, and call it a cent and a half per thousand. As to the statement of 3 men to 4 generators, the writer finds no difficulty in crediting this, as to his own personal knowledge 4 men do easily operate 6 Lowe generators.

Lime, and handling thereof, for purification of the fuel gas, may cost, as a high figure for a moderate-sized plant, two cents more per thousand. We have, then, for the probable total cost of putting purified fuel gas, by the Strong system, into the holders: $3.67 + 1.5 + 2 =$ say *seven cents and two-tenths per thousand feet.* Mr. Strong's

own estimate has been *eight cents*, which is evidently an entirely safe one.

This will produce gas, as shown above, of 9322° F. per pound; and as one pound of such gas at 60° F. contains (D. = .437) just about thirty cubic feet, one cubic foot contains 311° F. of thermic power. The writer has reasons, from facts on record, to anticipate that, for heating water up to boiling, suitable burners will utilize for us at least 75 per cent. of this, or say 230° F. per cubic foot. When heating air, as in warming houses, even a larger proportion will be made available.

Among the newer chapters in the history of what has been called the Fuel-Gas War, is a pamphlet, issued recently by an association of gas engineers of the first rank, entitled "The Waste of Energy in the Production of Water Gas." To this document are signed the names of the members of this society, by way of indorsement.

The writer, on having his attention lately called to this pamphlet, found with surprise its arguments to be based almost wholly on assumptions which do not bear examination. Of these fallacies only a few of the more important can be selected, as a complete discussion of this document would probably more than wear out your patience.

The manifesto of the Society of Gaslighting begins by promising strict and impartial scientific discussion, and proceeds then to the usual reiteration of hackneyed denunciations of water gas. First, it is *not new;* reference being made to the well-known English patent to the Kirkmans, of July, 1852, in ignorance of the practically identical previous patent to F. C. Hills, of January, 1852, and of the closely approximate patents of 1845 to William Pollard and John Constable, with the American patent to George Michiels, also of 1845. The Kirkman patent serves to introduce what seems to be a declaration of the intention of the Society of Gaslighting when it shall come that its members shall be forced to make water-gas, to do so without reference to existing patent-rights, assuming and asserting, in these words, that "the Kirkman process is that most largely used in this country," at the present day.

We next find reproduced the exploded assertion that water gas "was condemned and abandoned in France on account of it containing from 30 to 40 per cent. of the extremely poisonous carbonic oxide gas."

On the other hand, Dr. Adolphe Wurtz, one of the most eminent and learned of living chemists, wrote from Paris, June 12th, 1878, in

commenting upon an investigation of the writer of one of the improved processes, and the attacks that were made upon it, as follows: "The use of water-gas has never been prohibited in France, and if the numerous processes which have been indicated for its production have been abandoned, or have received only a restricted application, the cause is principally due to the circumstance that the technical and economical conditions of the production have, up to the present, been very unfavorable." He refers, of course, to the non-occurrence in France of indigenous materials suitable for this manufacture. He also says that "the danger (that is, of carbonic oxide in gas for domestic use), which could only produce ill results exceptionally and through fatality, has been exaggerated, and should not be taken into consideration." In reference to this part of the controversy, but two remarks will at present be offered.

Most gases, except pure air, are unfit for purposes of inhalation or respiration, and carbonic oxide shares this unfitness with others that are found in gas from gas coal. It is not, however, the purpose of the makers of fuel-gas to introduce an article for purposes of respiration. Nor is it intended to serve out to the public an *inodorous* gas, as has been averred, thus increasing the liability of accident. All fuel-gas made for household or other uses will be found to possess odors even more characteristic and alarming than that of gas-coal gas. As to those cases coming under the head of fatalities, such as blowing out the gas in a sleeping-room, these will occur with all gases. So, also, will men go to sleep upon railroad tracks; but this has not been deemed an argument against the railway system. So will coal-miners unlock and open their safety-lamps; but no one therefore demands that coal-mining be discouraged or discontinued. Moreover, carbonic oxide is actually now used, and far too largely and generally, for purposes of respiration; this being, in point of fact, one of those very lamentable defects of our present household organizations which *fuel-gas is destined wholly to cure.* The leakages and irregularities of our coal-stoves, heaters, and furnaces, which force us now so often to inhale carbonic oxide,—together with other gases, such as *sulphurous oxide,* a compound more poisonous, beyond all comparison, than carbonic oxide,—will be entirely avoided by the adoption of fuel-gas heaters of proper construction.

Again, it has been previously pointed out by the author, that risks from fire and explosion will be greatly less with carbonic oxide than with gas-coal gas, which latter contains from one-third to

one-half of marsh gas, or *fire-damp*, this being much the most explosive of all common combustible gases.

The document emanating from the Society of Gaslighting then proceeds to its main business, which is to prove that, in the conversion of carbon into fuel-gas, less than one-third of the thermic power of the carbon is left, more than two-thirds being necessarily wasted or dissipated altogether. This is a great advance on the earlier arguments of the opponents of fuel-gas, who only went so far as to assert that, as water, when unburned, must necessarily absorb just as much energy as its hydrogen engenders when burned, therefore the whole project must be unwise, unscientific, unpractical, and utopian. Not longer ago than 1873, technical journals, held in high and just esteem as educators of the public in technical matters, and of great circulation, used language indicating that this sort of thing was to be classed with perpetual motion and the like delusions. To illustrate, the following may be exhumed: "Notwithstanding the reiterated statement in the *Scientific American*, and other exponents of practical science, that it is impossible to utilize water as a fuel, because it takes as much heat to decompose it into oxygen and hydrogen as one can get from the recombustion of these gases, men continue to waste their time in inventing apparatus to accomplish it."

It appears to have been almost universally conceded that the undeniable proposition, founded on the conservation of energy, implied in the last paragraph, in enforcing the conclusion that *some expenditure* must needs accompany the manufacture of water gas, made it self-evident that all such schemes were unworthy the attention of the public and of practical men. The unbiased portion of the public has now begun, however, to comprehend that the existing practical conditions really, and indeed overwhelmingly, neutralize this seemingly sound and scientific argument; that the economy of use, the controllability, purity, cleanliness, healthfulness, safety, comfort, uniformity, indestructibility, reliability, easier confinement and storage, and other merits of fuel-gas will justify, if necessary, *considerable expenditure* in the making of it; and that the assumed application to this case of the grand truth of the conservation of energy involves a practical fallacy.

A new and great change of base on the part of the enemy appears, therefore, to have been decided upon; and in this pamphlet the attempt is deliberately made to obtain credence and currency for an asserted demonstration that the expense or "waste" in con-

verting the thermic energy of carbon into a gaseous form must needs be something like two-thirds of the raw material or solid fuel started with!

First. There is presented a *theoretical* computation.

Anthracite is stated to have a total theoretical thermic power per pound of 13,000° F. In reality, 14,000° is nearer, but it is probably not worth while to correct this now. Its *practical* value (for steam purposes, for example) is rated, however, as low as 6000° F.* For making fuel-gas, it is claimed that steam of as high a pressure as 100 pounds, say 7 atmospheres, is essential, the total heat of which is rated at 1153.4° F. per pound, which is low (1182.5° being about true, according to Trowbridge), but for simplicity this may also be admitted. 16 pounds of carbon and 24 pounds of water (as steam) are said to make 1000 cubic feet of equally mixed hydrogen and carbonic oxide, which is near enough for 60° F. Such mixture, in equal volumes, if it were obtainable, would weigh 40 pounds, and contain $2\frac{2}{3}$ pounds hydrogen and $37\frac{1}{3}$ pounds carbonic oxide. According to the admitted conservation-of-energy theory, this hydrogen, in burning (from 32° F.), engenders $62{,}500° \times 2.66 = 166{,}250°$. Such temperature must therefore be supplied by combustion of carbon, in order, theoretically, to unburn or decompose the water from which the hydrogen proceeded. It is, however, necessary to concede that the 16 pounds carbon, in burning to carbonic oxide with the oxygen of the steam, furnish $4450° \times 16 = 71{,}200°$; so that the amount of additional carbon, or rather, anthracite, required theoretically, at 13,000° per pound $= \frac{166{,}250° - 71{,}200}{13{,}000°} = 7.31$ pounds. The process of decomposition of steam by incandescent carbon is very strangely called *dissociation*. It may much more appropriately be called *combustion*, but we will not quarrel now with mere obscurities of language. So far, except fractional variations of data some of which may about balance each other, all is rational. And the result or product of the operation is 40 pounds of mixed hydrogen and carbonic oxide, but, theoretically, at the temperature of 32° *F.* An addition to the anthracite is, therefore, evidently necessary, determinable (with any degree of precision) only by experiment, representing what is necessary to heat the 40 pounds of gas, together with any excess of steam accompanying it, up to the temperature,

* For reasons apparent to an expert reader, they nevertheless rate *coke*—containing, as is well known, from 7 to 10 per cent. less carbon than good anthracite—at a practical value of 10,970° F. per pound.

above 60° F., at which they issue from the generator. This, at 500° — 60° = 440°, in the Strong system, *may* be (see below) something under a pound; say .9 pound of coal. Then $16\frac{1}{3}^6 + 7.31 + .9 = 26$ pounds of anthracite, in all.

This amount of anthracite, burned directly, has the theoretical value, $26 \times 13{,}000° = 338{,}300°$ F.; while 40 pounds of purified gas obtained therefrom, as above, in the Mount Vernon generator, have, according to the writer's analyses (see Table II), deducting, of course, the 15 per cent. (at least) of nitrogen by weight which is not derived from the anthracite, the value, $\frac{40^2}{40-(15 \times 4)} \times 7642°$ $= 359{,}633°$.

Here are two *theoretical* figures, which are directly comparable. Even if the value of 14,000° be assigned to the anthracite, we get then for total anthracite required, 25.4 pounds, and for its theoretical value, $25.4 \times 14{,}000° = 355{,}600°$ F., which is still some 4000° below the theoretical value of the Strong gas, theoretically obtainable therefrom. This curious fact is due, in some measure, to the considerable thermic value of the 5 per cent. of marsh gas present in the Strong gas, of which the Society of Gaslighting takes no account.

In the pamphlet, an addendum, ostensibly corresponding to the above, is made to the amount of anthracite theoretically required, in settling which "dissociation" is again mentioned, and to which the writer finds himself unable to attach any rational meaning whatever. The paragraph is as follows: "The temperature at which the dissociation of water takes place being 2192° F., according to Deville, the gas leaving the generator at this temperature, unless there be some method of utilizing the heat, carries off in heat, the temperature of the gas at the holder being 60° F.," an amount of the heat summing up 39,041° F. It seems to be asserted that the "temperature of dissociation" is that at which the gases *must* leave the generator. Now, while 2192° F. is less than half the temperature of dissociation *under constant volume*, according to estimates of Bunsen and Deville (4500° F., or higher), Deville obtained dissociation *under constant pressure* (that of the atmosphere) at some 1600° F. But it is wholly impossible to discern what we have to do with dissociation at all, or with any temperature, except the mean degree at which the products do actually leave the generator; of which, below.

The theoretical anthracite of the Society of Gaslighting adds up, including that which they insist on for purposes of dissociation, to

28.31 pounds. Even this, at 13,000°, is theoretically worth only 368,030° F., not yet much above the theoretical value of the fuel-gas yielded by it theoretically (as above, 359,633°).

Second. The Society of Gaslighting estimates the amount of anthracite "practically required to produce 1000 cubic feet" of fuel-gas.

The assertion is started with, that this case is one parallel with that of the waste of thermic energy in the steam-engine; rated in the pamphlet at nine-tenths of the fuel. There is no parallelism whatever between the two cases. Thus, where shall we discover, in the fuel-gas process, anything parallel to the loss of energy in exhaust steam? To consider the fanciful arguments brought in at this stage of their figuring will somewhat tax your time and patience. It is first asserted that, instead of 24 pounds of water, as steam, being needed to make 1000 feet or 40 pounds of gas, 50 pounds of steam at least are necessary, or an excess of 26 pounds, which must accompany the produced gases, carrying off an immense quantity of heat, which, as asserted, is necessarily wasted. Even were this true, it would be easy to save much of this, if at the temperature asserted, 2192°, or any other, by simply passing it through the flues of a boiler, and bringing it down to 300° F. or thereabout. But the writer has only to refer here to the record, which shows that in the Lowe process at Utica in 1875, the amount of this excess of steam in the products, as they come from the generator, was determined by him by quantitative analysis, as only 10,772 grains, or 1.6 pounds per 1000 feet; thus increasing the amount of steam to be made and used to only 25.6 pounds. Therefore, the amount of coal required to make this steam, which they state at $\frac{1153.4° \times 50}{6000°} = 9.61$ pounds, is really more nearly $\frac{1153.4° \times 25.6}{6000°} = 4.92$ pounds. In the Strong system it appears unlikely that any appreciable excess of steam could remain in the gaseous products, as these, after their formation, are subjected to a secondary operation of transmission downward through an incandescent mass of anthracite.

The 16 pounds of carbon is asserted to need 20 pounds of anthracite to supply it, an obvious exaggeration, 18 pounds being an ample allowance; if indeed, in the case of this figure, any allowance is called for, except for impurity in the anthracite, which would bring it below 17 pounds; 18 pounds will, however, be conceded. The temperature of the gas, as it leaves the generator, is, *at one stage* of the Lowe process, sometimes as high as 1200° F. (its *mean* temperature, however, being as yet undetermined), but in the Strong ex-

perimental apparatus the eduction-pipe does not reach more than 500° F., so far as the writer's observation has extended, or as he can learn by inquiry from others. In the Strong process, then, the possible loss arising from this source (assuming that no means are taken to save this residual heat) may be computed as follows:

Hydrogen, . .	2.66 pounds	× (its sp. heat) 3.4046	× (500°—32°)	= 4,238°
Carbonic oxide,	37.34 "	× " " .2479	× "	= 4,332°
Steam, . . .	1.60 "	× (its total heat at 500°) 1200°		= 1,920°
Possible loss of heat per 1000 cubic feet of gaseous products,				= 10,490°

To convert this into practical anthracite equivalent, the Society of Gaslighting would divide it by 6000, ignoring entirely the fact that this heat may fairly be all regarded as *recovered heat* of the products of combustion, recovered by the action of the regenerative appendage used in both the Strong and Lowe systems. Even if this be not insisted on fully, as the writer believes justifiable, yet the divisor 6000 is here of course absurdly inapplicable, and the lowest divisor that could be rationally adopted is the full assumed theoretical value, 13,000°. This makes the anthracite consumption due to residual heat $= \frac{10,490}{13,000} = .8$ pound; a figure to be substituted for the total figure ciphered out by the Society of Gaslighting, which is 8.35 pounds. Our total estimate of anthracite consumption in making 1000 cubic feet of Strong fuel-gas is then: $4.95 + 18 + .8 = 33.7$ pounds. This figure may be usefully compared with the best actual result on record of the very imperfect experimental plant at Mount Vernon = 35 pounds; two-thirds of which were screenings.

According to the Society of Gaslighting, such weight of anthracite is practically worth $33.7 \times 6000° = 202,200°$ F., while 40 pounds Strong gas, made therefrom, as previously computed, is worth, *theoretically*, 359,633° F.; difference = 157,433°.

It remains to be seen how large a percentage of this total theoretical value of 1000 feet of fuel-gas will be available when this gas is used for heating, cooking, motor, metallurgical, and other uses. It may be pointed out that only 55 per cent. of utilization = 195,982° F., pretty nearly obliterates the "waste of energy" of the Society of Gaslighting, when its own valuation of anthracite coal is adopted. Now it happens that 55 per cent. is just the proportion of the theoretical heat of gas-coal gases, stated by a distinguished gas chemist, Dr. Wallace, of Glasgow, the gas examiner of that city, to have been

recently obtained by him in experiments in heating water, without the use of Bunsen burners. Moreover, our own very ingenious and industrious gas expert, Mr. Goodwin, of Philadelphia, has recently published experiments showing that, under the conditions of the Bunsen burner, some 25 per cent. less gas will do as much work in heating water below boiling, as when ordinarily burned.

Before leaving the pamphlet of the Society of Gaslighting, it is necessary to refer to the citation therein of some experiments by E. Vanderpool, Esq., and Dr. A. F. Schuessler, who made, as they state, a mixture of hydrogen 65 and carbonic oxide 35 per cent., and give a determination of its thermic value as 136.6° F. per cubic foot. As *pure* hydrogen and carbonic oxide, in these proportions, must possess, at 60° F., a value per cubic foot of 324.5° F., this experimental result shows a utilization of but 42 per cent. of the total power. This, the pamphlet pointedly remarks, "may be taken as practically reliable." But, as no analytical or other evidence is presented of the absence of foreign inert gases from the mixture made, this surprisingly low result certainly justifies the presentment of the hypothesis of the ingress of such inert gases, in some such way as to evade the vigilance of these gentlemen. Few of the methods for the preparation of the two gases operated on yield products of even approximate purity.

The same experts also give a determination of the thermic value of gas-coal gas ("ordinary 16-candle gas") as 318° F. per foot; while in another part of the pamphlet (what is presumably) the same gas is rated at 660° F.; estimates of its economy as compared with so-called water-gas being based on the latter figure, to the neglect of the former, arrived at by actual experiment.

The writer feels compelled also to refer to the fact that, in quoting the figures of Sarnstrom, from the correspondence of George S. Dwight, Esq., from Stockholm, Sweden, as given in the *Engineering and Mining Journal* of August 30th, 1879, the writer or writers of the pamphlet would seem to have made an oversight, or selected figures to suit their argument. The result of comparing the Strong gas, as made at Stockholm, with gas-coal gas, are given by Mr. Dwight in three different forms, two of which agree with each other, and not with the third, which latter inferentially, therefore, involves some miscalculation. The two which agree give the fuel-gas a value of more than half that of the special gas-coal gas compared, while the other (the one selected and used in the society's pamphlet) makes the gas-coal gas 2.2 times as powerful.

The following wonderful statement from this pamphlet of the Society of Gaslighting may help to account for the mental obliquities which must have contributed to the fallacious reasoning and untenable conclusions found therein: "A glass globe exhausted of air, under constant pressure and temperature, can be filled with the vapor of water, and there is still room for the globe full of alcohol vapor, and then there is still room for the globe full of ether vapor—and we might go still further." We might, in all humility, inquire what the gas analysts are to do, now that this new law of nature, subverting all their processes, has been discovered by the Society of Gaslighting?

A subsequent paper will be submitted on the relations of fuel-gas and the Strong system to *illuminating gas,* and to the closely related Lowe system of making the latter, in which facts and statistics of great public interest, now in the course of collection and preparation, will be brought forward.

No. 25 London Terrace, New York, February, 1880.

THE CLAIBORNE GROUP AND ITS REMARKABLE FOSSILS.

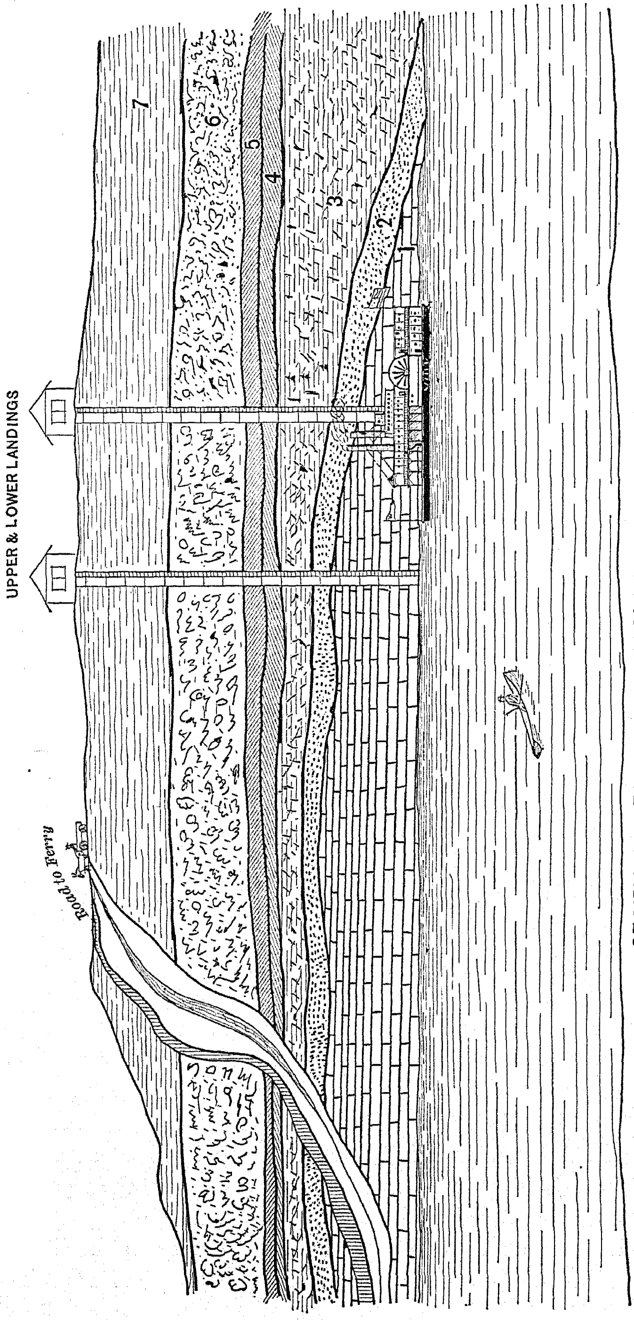
BY PROFESSOR P. H. MELL, JR., AUBURN, ALABAMA.

The little village, from which this formation receives its name, is situated on a bluff of the Alabama River, 175 feet above water level. This bluff is a portion of high table land that begins in the western portion of Choctaw and extends through Clarke and Monroe counties. We find points on this ridge more than 100 feet higher than Tuscaloosa, and nearly as high as the Tennessee River at Tuscumbia. Through this the Alabama River cuts its way, exposing a beautiful cross-section of the strata, of which the accompanying sketch may give some idea.

The fossils are found about 75 feet below the surface and 100 feet above the level of the river. There are seven distinct strata.

1. The lowest, extending from the water's edge up the bluff 80 feet, is a soft calcareous rock containing about 15 per cent. of carbonate of lime, with occasional specks of mica. In this a few *Ostrea, Turritellæ,* and *Flustræ* were found, but so friable as to fall to pieces when handled.

2. Next comes a stratum varying from 2 to 8 feet, consisting of



SECTION OF FORMATION AT CLAIBORNE, ALABAMA.
Bluff 175 to 200 feet.

a more compact calcareous rock containing about 30 per cent. of carbonate of lime. In this were found the fine specimens of *Ostrea* accompanying this paper. On close examination numerous grains of greensand were detected scattered through the mass.

3. Above this last is a stratum, the thickest part of which is over 30 feet, narrowing at places to less than 4 feet. From this the largest proportion of the fossils were obtained. It is exceedingly rich in animal remains, containing more than 250 distinct species of shells alone, while a variety of bones, shark's teeth, etc., have also been found in great abundance. "The bed is composed of loose quartzose, brownish sand, the grains of which are small and angular. The most perfect specimens can, therefore, be removed with ease in a perfect state." The fossils from this stratum are most wonderful on account of their perfect preservation of form and color. One would naturally suppose that they were gathered from the present seashore. The animal seems to have just left them.

4. Above this layer is a thin friable rock, easily separating into irregular pieces, composed of light and dark greensand. The grains of sand do not present angles, but are rounded and cemented by carbonate of lime.

5. A thin layer of sand and shells which are badly decomposed.

6. The next in order is a stratum of rotten limestone about 45 feet thick.

7. On top of this is deposited the diluvium strata.

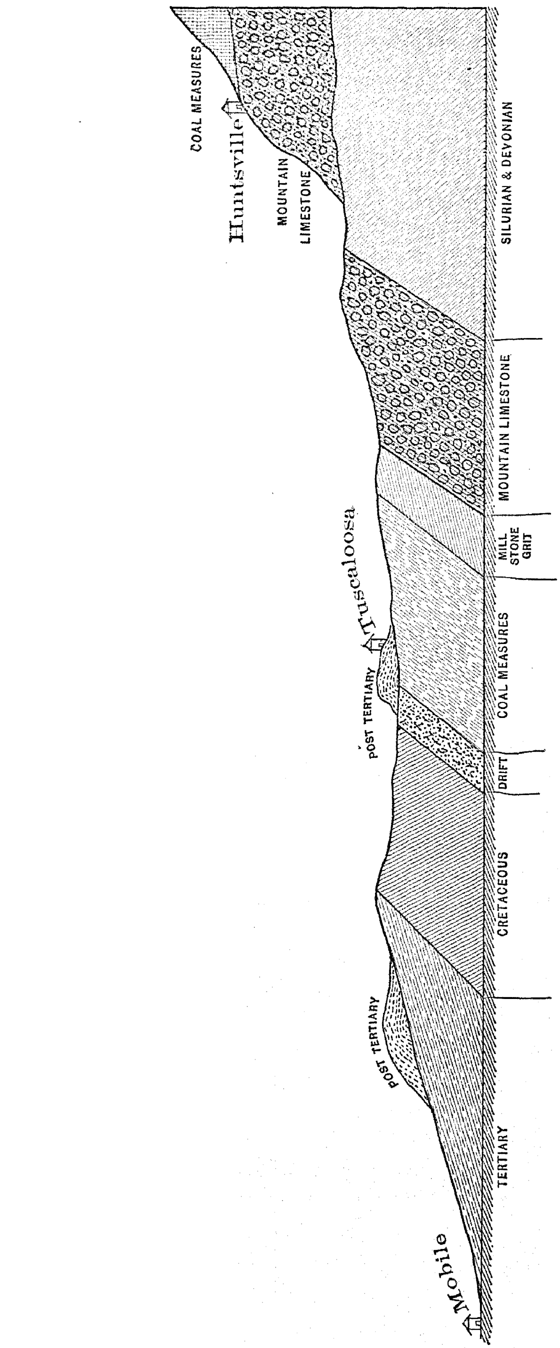
In connection with this formation at Claiborne, but a little farther south, is an extensive deposit of the remains of the *Zeuglodon cetoides*, the whale of the tertiary seas. These fossils are found in vast numbers around Gainestown, Clarke County. The deposit trends in a southwest direction. The vertebræ, particularly, are in such vast quantities that some farmers use them for building fences. They are exceedingly large, some measuring fully 12 inches in diameter and 15 to 18 inches in length. I have one now in my museum of these dimensions. The specimen I send for the collection for the German Geological Institute is rather smaller than the majority found at this locality, but it gives, however, a fair representation of the character of the remains. Lyell says that the vertebral column of one of these huge animals has been found in place nearly 70 feet long, but there is every reason to believe that some of these monsters must have attained the enormous length of 100 feet.

These fossils seem to belong to but one locality in the State, or at least the present developments have exposed no other deposit. This

is somewhat singular when we consider that the Tertiary covers such a wide area of country. I have but one explanation to give. It is a well-known fact that the recent formations were deposited by the waters of the then existing ocean, and also by the great streams flowing across the carboniferous, silurian, and cretaceous countries. The irregularity of this deposition at certain points is evidently the result of currents and eddies in this great expanse of water. Such being the case other beds of fossils similar to the above may be found by cutting through the upper stratifications. There must have been depressions and elevations on the bottom of the then existing ocean, as the present contour of the land seems to indicate; and the present outcrop of *Zeuglodon* fossils may have been an elevated place in the sea.

This portion of Alabama is of special interest to the geologist as presenting features not to be found in any other portion of the world. Passing for the first time from the tertiary formation of Georgia to that of Alabama, one would be struck by the marked contrast between the two sections. In the first State the country is comparatively level and regular in form, while in the latter the surface is broken and covered by innumerable hills and ridges, some of which are quite lofty. The Claiborne ridge is a good illustration of this feature of the country. A reason for this great difference between the two sections of the same formation would be quite difficult to give unless a thorough knowledge is first obtained of those older deposits covering the northern portion of the State. By a glance at the geological map of Georgia it will be noticed that the Tertiary borders on the Drift, a narrow strip that separates it from the metamorphic region. In Alabama, however, a wide belt of Cretaceous juts up between the Drift and Tertiary, as is seen in the accompanying sketch.

The Cretaceous first makes its appearance on Georgia soil not far east of the city of Columbus. It sweeps around towards Selma, Alabama, beyond which point it takes a northwest course into Mississippi. Now it is my opinion the Tertiary of Alabama was almost, if not entirely, formed from the Cretaceous, and the building agent was *ice*. I have been very much strengthened in this view after carefully studying the other formations of the State. Just prior to the time that the Tertiary was lifted up above the waters the geographical features of Alabama were very favorable for the transportation of ice, and every indication seem to point to the fact that the present Coosa and Warrior rivers were the great channels



GENERAL SECTION N. & S., SHOWING THE DISPOSITION OF STRATA
(after Tuomey.)

through which the glaciers from the north reached the State. Scattered over the Cretaceous are certain isolated points elevated considerably above the present level. Capping these islands, as it were, is a compact sandstone, underlying which are the cretaceous marls and limestones. This sandstone is found only on these elevated points, and associated with it are quantities of water-worn pebbles. The remainder of the Cretaceous is in the form of a huge trough, the portion joining the Tertiary being much higher than the central part.

Now these high points, all containing the same stratum of sandstone and water-worn pebbles, seem to indicate that the Cretaceous must have been at some past age entirely overlaid by this sandstone. What then has placed it in its present peculiar form? The force, above mentioned, must have been a powerful agent in this transformation. While this grinding and wearing away of the Cretaceous was taking place the Tertiary was still covered by water, but not to a great depth. As this material was thrown into the sea it was caught up by the many currents that existed and spread over the wide expanse of bottom.

I am also of the opinion that several of the strata displayed at Claiborne, which contain but few fossils, must have been deposited in a short space of time by the great influx of ice-bearing sediment, while on the other hand, the stratification containing the vast numbers of fossils must have been deposited by slow and quiet means, a great time elapsing before the deposition was completed. The fossils were not brought in by a great and violent current, but were placed there by the animals themselves after living the time allotted them by nature. This opinion is confirmed when we notice how well preserved is their delicate form and color. I have found but the smallest proportion that are anyway water-worn.

Now for a practical deduction from the facts and surmises given above. Although I have made but one trip into the Tertiary of Alabama, I have, nevertheless, been forcibly struck with the immense accumulation of animal remains—bones of huge sea monsters and teeth of enormous sharks, all mingled together in a heterogeneous mass, and spread over the entire southwestern part of the State. Scattered among these remains are small rounded grains of greensand, similar to that variety in New Jersey that has been of such great benefit to the farmer in the production of potash and phosphoric acid. Is it not reasonable to suppose that where two such associates, as bones yielding phosphoric acid and greensand

producing potash are found, extensive phosphate beds, resembling those at Charleston, S. C., may at some future day be discovered if diligent search be made? At least I am inclined to predict that a few years hence Alabama will be carrying on extensive mining operations for this valuable fertilizer.

With a few directions as to the best route to Claiborne I will close. If it is desired to study the cross-sections of both the Cretaceous and Tertiary formations the best plan will be to go directly to Montgomery, Alabama, and board the river steamer that leaves that city about three times a week. It will be necessary, however, for the tourist to be provided with an extra amount of patience before engaging a berth on one of these vessels, especially during the cotton-trading season. The speed is exceedingly slow,—not more than ten miles per hour,—and the stoppages are frequent, wherever a bale of cotton can be obtained. But, on the other hand, the accommodations are very good, and the officials particularly polite and attentive. Such a trip as this would be of interest to one who loves to study human nature in all its varied forms, for certainly the types are of many kinds. It generally takes about two days and two nights to make the trip from Montgomery to Claiborne, the distance being 382 miles. The fare between these two points is $6.00.

If, however, there is but limited time in which to make the trip, the advisable plan will be to pass through Montgomery and go immediately to Mobile, *via* the Montgomery and Mobile Railroad, and take the boat at this last place for Claiborne. The distance from Mobile to Claiborne is only 100 miles, and the fare about $1.50. Board can be obtained in Claiborne at the rate of $1.00 per day, backed up by good country hospitality. Of course the tourist will not fail to avail himself of the opportunity for visiting the deposit of *Zeuglodon cetoides* fossils, and a sojourn of a day or two in the neighborhood of Gainestown, Clarke County, will be required; after which he may continue his trip towards Claiborne by the next steamer.

The following is a list of the fossils found at Claiborne. Those in italics I have sent with this paper as a contribution to the collection for the German Geological Institute. I hope to be able to make the collection complete in the course of two or three months. Where no name is given Lea is supposed to be the authority.

Lunulits Bouéi.
" *Duclosii.*
Orbitolites interstitia.
" discoidea.
Turbinolia Maclurii.
" *Stokesii.*
" *Goldfussii.*
" *nana.*

Turbinolia pharetra.
" *Caulifera* (*Con.*).
Siliquaria Claibornensis.
Dentalium alternatum.
" *turritum.*
" biformis (*T*).
" thalloides (*Con.*).
Spirorbis tubanella.
Serpula ornata.
Teredo simplex.
Solecurtus Blainvillii.
Anatina Claibornensis.
Mactra dentata.
" Grayi.
" pygmæa.
Corbula Alabamiensis.
" Murchisonii.
" *gibbosa.*
" *compressa.*
" oniscus (?).
" nassuta (?).
Byssomia petricoloides.
Egeria rotunda.
" *inflata.*
" *nitens.*
" *triangulata.*
" *Bucklandii.*
" *subtrigonia.*
" *venereformis.*
" *ovalis.*
Egeria plana.
" *nana.*
Lucina compressa.
" rotunda.
" cornuta.
" *impressa.*
" papyracea.
" *lunata.*
" carinifera (*Con.*).
" pandata (*Con.*).
Gratelupia Monlinsii.
Astarte recurva.
" *Nicklinsii.*
" *sulcata.*
" parva.
" minor.
" *minutissima.*
" Tellinoides (*Con.*).
Cytherea globosa.
" *comis.*
" *Hydii.*
" *subcrassa.*
Cytherea trigoniata.
" *minima.*
" perovata (*Con*).
" Poulsoni (*Con.*).
" Nuttalli (*Con*).
" Sayana (*Con.*).
" albarea (*Say.*).
" æquorea (?).
" eversa (*Con.*).
Venericardia transversa.
" *Sillimani.*
" *rotunda.*
" *parva.*
" *planicosta* (*Lam.*).
Hippagus isocardioides.
Myoparo costatus.
Arca rhomboidella.
Pectunculus Broderipii.
" *minor.*
" *deltoideus.*
" *ellipsis.*
" *obliqua.*
" stamineus (*Con.*).
" sp. (?).
Nucula Sedgewickii.
" *ovula.*
" *pectuncularis.*
" *Brogniarti.*
" media.
" pulcherrima.
" *plicata.*
" magna.
" carinifera.
" plana.
" semen.
" magnifica (*Con.*).
Avicula Claibornensis.
Pecten Deshaysii.
" Lyelli.
Plicatula Mantellii.
Ostrea Semilunata.
" *divaricata.*
" *Alabamiensis.*
" lingua canis.
" pincerna.
" emarginata (*T.*).
" compressa-rostrea (*Say*).
Fissurella Claibornensis.
Hipponix pygmæa.
Infundibulum trochiformis.
Crepidula cornu-arietes.
" lirata (*Con.*).

Bulla St. Hilairii.
" Dekayi.
Pasithea secale.
" *notata.*
" lugubris.
" aciculata.
" striata.
" sulcata.
" *umbilicata.*
" guttula.
" Claibornensis.
Natica striata.
" parva.
" minor.
" *minima.*
" *gibbosa.*
" *semilunata.*
" magno-umbilicata.
" *mamma.*
Acteon punctatus.
" *lineatus.*
" *elevatus.*
" melanellus.
" striatus.
" pygmæus.
pomilius (Con.).
Scalaria planulata.
" carinata.
" quinquefasciata.
Delphinula plana.
Delphinula depressa.
Solarium bilineatum.
" *Henrici.*
" *ornatum.*
" *elegans.*
" *cancellatum.*
" granulatum.
" elaboratum (*Con.*).
" alveatum (*Con.*).
Orbis rotella.
Planaria nitens.
Turbo naticoides.
" *nitens.*
" lineata.
Tuba striata.
" alternata.
" sulcata.
Turritella carinata.
" *lineata.*
" *imbricata (Lam.).*
" mortoni (*Con.*).
" obruta (*Con.*).
Turritella Cathedralis (Brongn).
Cerithium striatum.
Pleurotoma cælata.
" *Lonsdalii.*
" Sayi.
" monilifera.
" Baumontii.
" *Desnoyersii.*
" Hæninghausii.
" *rugosa.*
" obliqua.
" Childrenii.
" *Lesueurii.*
" acuti-rostra (*Con.*).
" sp. (?).
Cancellaria Babylonica.
" multiplicata.
" *plicata.*
" sculptura.
" tessellata.
" elevata.
" costata.
" parva.
" gemmata (*Con.*).
Fasciolaria plicata.
" *elevata.*
Fusus pulcher.
" *Mortonii.*
" decussatus.
" *bicarinatus.*
" *venustus.*
" *crebissimus.*
" *magnocostatus.*
" Delabechii.
" *ornatus.*
" acutus.
" Conybearii.
" *nanus.*
" *Fittonii.*
" parvus.
" minor.
" *Taitii.*
" *sp.* (?).
" spiniger (*Con.*).
" salebrosus (*Con.*).
" papillatus (*Con.*).
Pyrula cancellata.
" elegantissima.
" Smithii.
Murex alternata.
Rostellaria Lamarckii.
" Cuvieri.

Rostellaria velata (*Con.*).
" *laqueata* (*Con.*).
" *alveata* (*Con.*).
Monoceros pyruloides.
" *fusiformis.*
" sulcatum.
" *vetustus* (*Con.*).
" armigerus (*Con*).
Buccinum Sowerbii.
Nassa cancellata.
Terebra gracilis.
" costata.
" *venusta.*
Mitra lineata.
" minima.
" *fusoides.*
" Flemingii.
" Humboldtii.
Voluta Defrancii.
" *gracilis.*
" *parva.*
" Vanuxemi.
" *sp.* (?).
" striata.
" *sp.* (?).
" *Parkinsonii.*
" *Cooperii.*
" *Sayana* (*Con.*).
" Tuomeyi (*Con.*).
" *petrosa* (*Con.*).
Marginella anatina.
" columba.
" *Crassilabra.*
" plicata.
" *semen.*
" *ovata.*
" *incurva.*
" biplicata.
" larvata (*Con.*).
Anolax gigantea.
" plicata.

Oliva constricta.
" *gracilis.*
" *Greenoughii.*
" *dubia.*
" *Phillipsii.*
" minima.
" *sp.* (?).
" Alabamiensis (*Con.*).
Montoptygma Alabamiensis.
" *elegans.*
Conus Clairbornensis.
Genus (?) *Species* (?)
Genus (?) *Species* (?)
Unknown.
Bone of a fish.
Spine of a fish.
Shark's tooth.
Turbinella pyruloides (*Con.*).
Sigaretus sp. (?).
Pholas Roperiana (*Con.*).
Modiola Ducateli (*Con.*).
Melongena alveata (*Con.*).
Emarginula arata (?).
Crassatella Mississippiensis (*Con.*).
" *alta* (*Con.*).
" *protexta* (*Con.*).
Zeuglodon cetoides.
Corbis lamellosa (?).
Conus sauridens (*Con.*).
Cassis Taitei (*Con.*).
Cardium Vicksburgense (*Con.*).
" diversum (*Con.*).
" Nicoleti (*Con.*).
Cardita plani-costa (*Con.*).
" alti-costa (*Con.*).
" rotunda (*Con.*).
" densata (*Con.*).
Ancillaria subglobosa (*Con.*).
" staminea (*Con.*).
" scampa (*Con.*).

THE SUCCESSFUL MANUFACTURE OF PRESSED FUEL AT PORT RICHMOND, PHILADELPHIA, PA.

BY E. F. LOISEAU, PHILADELPHIA.

In a paper on the manufacture of artificial fuel, read at the Philadelphia meeting of February, 1873, I enumerated the difficulties which I had to overcome before succeeding in the mixing of coal-dust and clay, the compressing of the same mixture, and the water-proofing of the lumps. The drying of the lumps, after leaving the press, was the remaining difficulty, and it was expected that a plan devised by Dr. Charles M. Cresson, of Philadelphia, would enable us to dry the fuel as rapidly as it was moulded, and that a continuous production could in that way be obtained.

The company was reorganized. The works were purchased by the new company at an assignee's sale, and the oven was modified, according to Dr. Cresson's plan.

Anticipating a possible failure, I had prepared a plan by which I expected to be able to demonstrate that anthracite coal-dust mixed with pitch, could be manufactured with our present machinery slightly modified; so that after all, if we were compelled to give up the attempt to make fuel for domestic use, there was a possibility of succeeding in the manufacture of a good steam-fuel.

The plan suggested by Dr. Cresson for drying the pressed lumps of coal-dust cemented with clay, did not work as well as we expected. It enabled us to dry more fuel than we did before, but it could not be made to dry more than one-half of the lumps produced by the press. The plan was abandoned, and I was authorized to experiment with coal-dust and coal-tar pitch.

The cement which is used in Europe to conglomerate coal-dust is usually dry pitch, which is prepared by separating from the tar, at a temperature of 572° Fahrenheit, the volatile matters which it contains. Some manufacturers, however, employ crude tar, others, a rich tar, which has been cleared of 25 per cent. of its volatile substances, by heating it to 392° Fahrenheit. But with common tar very weak fuels are obtained, which do not burn well, and give out a strong smell and a great deal of smoke; it is also necessary to subject them to a baking process, in order to solidify them, and to eliminate the more volatile of the materials contained. This operation of course requires a special plant, the cost of which increases sensibly the price of manufacture, without counting the products

which are lost, which have an industrial value. The crude coal-tar is also much inferior to the dry pitch, which can be broken and even pulverized when cold, and be thoroughly mixed with the coal-dust. This produces briquettes that give off very little smell.

The mixing of the coal-dust and pitch is usually carried on in a vertical cylinder, into which the coal-dust and pitch are charged continuously and automatically. These subtances are heated gradually in the cylinder or mixer by jets of steam which are discharged upon them from all sides; they are then triturated and amalgamated by a series of blades fixed on a vertical shaft. Arriving at the bottom of the cylinder, the materials are discharged in a pasty condition, through openings, from which they are placed or conveyed to the moulds.

In order to obtain a good lump from this paste, the pressure must be at least 3000℔ per square inch, and in certain cases, with hard or lean coal, it is necessary to increase this by 50 per cent. This heavy pressure is required by the nature of the paste, in order to expel the water which it contains, and to bring it to a compact condition. In European mixers the steam injected into the materials escapes with difficulty and condenses rapidly, hence the moisture in the mixture, which is only expelled by strong pressure.

When steam is injected through perforations into the materials to be mixed it loses in reality its pressure, that is, the tendency to push asunder the sides of its containing vessel; but at the same time it produces a temperature corresponding to a considerable pressure. Steam gives up first its latent heat, and then, after suffering condensation, a portion of its free heat corresponding to the difference of temperature, and the mass thus becomes continually heated. This, however, requires time, and it occurred to me that if I could dry the coal-dust first, bring the same to a certain degree of heat, and mix it with coal-tar pitch in a molten state, I would obtain more rapidly a plastic mixture which could be moulded by the same rollers used previously to mould the mixture of coal-dust and clay.

I was well aware that my mixer was not the right apparatus to mix rapidly coal-dust and melted pitch, but I had seen at work a mixer invented by Mr. August Dietz, of Philadelphia, for the mixing of sand and asphaltum for paving purposes, and I had no doubt that it could be modified to answer my purpose.

Before obtaining the means to make the required alterations in the plant, I had to demonstrate the possibility of making the fuel in

this way. I made the demonstration in a very primitive way. I hired two men engaged in the tar and gravel-roofing business, and had them melt the pitch in the yard and hoist it up in buckets, from which I dipped the pitch with a gallon measure, and emptied it into the mixer. A certain quantity of coal-dust previously heated, had before this been discharged into the mixer. In the bottom of the mixer I had placed a steam pipe, 1 inch in diameter, with perforations of $\frac{1}{8}$ inch, through which I injected steam into the materials until they were brought to a plastic condition, when I gradually discharged them into the hopper of the press, and moulded the same without difficulty.

The moulding rollers are hollow, so as to enable us to warm them by steam. As I had no steam connections made, in order to prevent the adhesion of the materials to the rollers the moulds were lubricated by means of two tin pans, filled with water, placed underneath each roller, and in which it revolved to a certain depth.

The lumps were very hard; the demonstration seemed to be conclusive; at least it appeared so to one of our stockholders, who offered to make the required alterations at his own risk if he was allowed to try a mixer which he had devised, and which, he thought, would answer my purpose as well as Dietz's mixer. The attempt was not a successful one, and as our means were nearly exhausted, I had but a poor chance of carrying out my ideas, when another stockholder came in who approved my plans, and offered to apply them, on certain terms and conditions, which were accepted by the company.

There is a rule attributed to Bacon which says: "Begin with observation, go on with experiments, and supported by both, try to find a law and a cause." I tried my best to apply that rule. The man who is experimenting, and wants to have absolute facts to work upon, is often made to doubt his own sagacity and capability, for he must often change his course of action by reason of deductions drawn from experiments. It so happened with me. I had carefully planned with Mr. Dietz all the details of his mixing machine, in order to adapt it for our purpose. Still I had lost sight of one essential point, and that was to keep the materials, when mixed and brought to a plastic condition, in a hot condition in a close conveyer instead of an open one, as we have now. The pitch acquires its cementing properties from 170° to 212° Fahrenheit; below 170° degrees it loses them. When exposed to the atmosphere, the mixture chills gradually, and when the pitch coating of the par-

ticles of coal is chilled it prevents the perfect adhesion of the particles under pressure. While the pressed lumps are still warm their surfaces are smooth, and the chilled particles apparently adhere, but when the lumps are cooled, the rubbing of one lump against another sets loose the chilled particles which accumulate and create dust again in the coal-pockets in the carts, and in the cellars of the customers. This defect, however, can be easily remedied by replacing the open conveyer under the mixer by a closed one, and heating the moulding rollers with steam.

In Dietz's mixer are two horizontal shafts, to which are clamped a series of blades placed at opposite angles, and which make 35 revolutions in a minute. When the materials are mixed they are dropped into the conveyer underneath, through apertures in the bottom of the mixer, which are opened and closed by means of sliding doors operated by a lever. In this conveyer the materials are also carried forward towards the hopper of the press, by blades placed at the same angle on two horizontal shafts, but they make only 3½ revolutions per minute. With this mixer a quantity of materials, weighing a little over 1000℔, is mixed and brought to a plastic condition, ready to be moulded, in the short space of 2½ minutes.

The coal and the pitch are both measured, and the proportions are 9 per cent. of pitch to 91 per cent. of coal-dust.

The moulding press is composed of two rollers geared together, on the periphery of which are milled out a series of semi-oval cavities, connected with one another, in order to facilitate the dropping of the lumps from the moulds on an endless belt placed underneath.

The efficacy of moulding rollers is not accidental or arbitrary, but is governed by certain rules which may be determined on mathematical principles, if not with perfect exactitude, at least with a tolerable degree of accuracy. Moulding rollers accomplish the compression of materials more by a squeezing or bruising action. They possess the great advantage of squeezing the materials so that the feed is only a short time between the rollers. This advantage is a very important one, and it will not be surprising if rollers, as a matter of fact, are destined hereafter to play a great part in the manufacture of artificial fuel.

If we follow the materials in their passage through two rotating rollers, we find that they begin to adhere at a certain point, depending partly on the dimensions of the rollers and partly on the size of the lump. The particles of coal coated with pitch receive no pressure at the first point of contact from the face of the rollers, but

from the drawing-in action of the two revolving rollers. The squeezing pressure which is thus exerted on the materials is produced entirely by the gear of the rollers, because, through the rotating motion, the plastic mixture is drawn into a gradually decreasing compass, and must be highly compressed and moulded. This reduction takes place regularly, both rollers possessing an equal speed. The speed being equal, the product leaves the rollers in the shape given by the moulds.

If the arrangement of the compressing rollers is such that they may be approached to one another at will, by means of springs, the first result must be a diminution of the amount of power required, in comparison with the rollers with fixed pressure. The feeding of the materials will also be more regular, and the danger of breakages from pieces of iron, stones, etc., which are often found in the coal-dust, will be avoided, the springs yielding to allow the passage of these foreign substances through the rollers. It is to be regretted that our rollers are brought together by means of screws, instead of springs.

The great difficulty is the regulating of the feed. Rollers of large diameter draw in the feed better than those of a smaller diameter. The feed ought to enter under the regulating diaphragm, along the whole length of the rollers in an even stream; still this cannot always be the case, because the stream of materials is not even. A certain friction takes place between the particles of coal and pitch, because the proportion of pressure on the particles of the feed in the middle varies from the pressure exerted on the particles next the rollers, the latter being more compressed, and sometimes crushed. The entry of the feed, should therefore not be forced, for in this case, either a portion of it will pass through the rollers not sufficiently compressed, or a stronger pressure will have to be employed, which would alter the result desired, and would produce lumps sufficiently compact to resist rough handling without breaking, but not sufficiently porous to insure free combustion, without a blast or a strong draft.

The greatest difficulties experienced in the moulding of the coal and pitch were to obtain a regular feed of materials, and to prevent the accumulation of materials which solidified in the hopper of the press. These accumulations prevented also the regular delivery of the materials between the rollers. I succeeded in overcoming these difficulties by a very simple contrivance which works perfectly.

The coal-dust is dried and heated by two sets of four revolving

drums, which answer well enough in dry weather, but when the coal is very wet we have some difficulty, and we are unable to dry and warm a quantity of coal sufficient to keep the mixer and the press running. This defect, however, can be easily remedied by increasing the size of the outlets for the escape of the moisture evaporated from the coal.

The defects of the present plant could have been corrected long ago, had I had the opportunity of carrying out my ideas. Through force of circumstances I was compelled to allow others to try plans of their own. The result was expensive, unsatisfactory, and unsuccessful experiments, the legitimate outgrowth of which was disappointment, disagreement, loss of time, of money, and of production. At last, however, I was allowed to have my own way, and the result was a success, although obtained with imperfect means.

The coal was placed in the market by myself, and I introduced it from the start for domestic use. It was supposed that the smoke and the strong smell of the burning pitch would be a serious objection to its use, but by careful instructions given to the customers, the inconvenience from the smell and smoke was hardly perceptible to those who followed instructions.

While experimenting with the fuel in different heating apparatus, I ascertained that when the lumps were but half consumed, if the poker was handled roughly, the particles of coal would disintegrate and would fall, unconsumed, through the grate-bars into the ash-pan, seemingly increasing the quantity of ashes, but in reality losing the heating power of the unconsumed coal. This was caused when the lumps were red-hot to a depth of about a quarter of an inch. Each lump would then become, so to say, a small retort. The pitch which held the particles of coal together, in the centre of the lump, would gradually be drawn through the red-hot crust of the lump, and be consumed, and when the lump itself was partly burnt, and reduced to about one-third of its volume, there was not sufficient pitch left in the nucleus to keep the particles of coal together until they were consumed.

In order to remedy this very serious defect I mixed with the anthracite coal-dust about 8 per cent. of powdered bituminous coal. The result was a better fuel, which did not disintegrate, coked in the fire, and was almost entirely consumed, leaving but a small quantity of ashes, when compared with the fuel made from anthracite without the addition of bituminous coal.

This last fuel has found a ready market. It ignites readily, lasts

as long as the ordinary anthracite coal, and it does not clinker. A good many of those who have tried it do not wish any other, and they send in new orders whenever their supply is exhausted.

It has been the main object of all inventors of machinery for the manufacture of artificial fuel, to obtain a large production in lumps of a small size. It is easy to obtain a large production in lumps of a large size, and no better machine has yet been devised to obtain a large production than that described by Dr. Grimshaw in the *Journal of the Franklin Institute*, of September, 1879, and which is manufactured in France, by the Société Nouvelle des Forges et Chantiers de la Méditerranée. The production of a double machine, of the smallest size, does not exceed 96 tons in 24 hours, in lumps weighing very near 3℔. My press will manufacture in one hour, 13 tons of lumps weighing only 2½ ozs. each. These lumps require no drying or baking. They are conveyed to a screen in eight minutes, and that time is sufficient to cool the lumps. They are then ready for delivery.

The pressed fuel would be much improved if the coal-dust was previously washed, and in the erection of new works, it will be essential to provide washing apparatus for that purpose.

The difficulty now seems to be to secure a sufficient supply of coal-dust at the shipping-points; and as there is a market for pea and dust, the coal companies do not feel inclined to dispose in our favor of the dust proper, so as to enable us to manufacture a fuel which would compete with their own coal. The successful manufacture of the pressed fuel, being, however, a demonstrated fact, it will evidently be in the interest of the large companies to erect machinery to utilize the coal-dust, instead of piling it up around the mines. Whether the manufacture of the pressed fuel is carried on by us or by the coal companies, the community at large will be benefited by the utilization of coal-dust, which was considered until recently, a worthless material.

I have struggled during twelve years to obtain this result. I persevered under the most trying circumstances, having to overcome financial as well as mechanical difficulties. I am satisfied now that very little remains to be accomplished, in order to make the manufacture of pressed fuel from coal-dust one of the most important industries of Pennsylvania.

PHILADELPHIA, February 13th, 1880.

NOTES ON THE SIEMENS DIRECT PROCESS.

BY A. L. HOLLEY, C.E., LL.D., NEW YORK CITY.

THERE is a growing demand for pure and cheap material for fine open-hearth steel; a material not only very free from phosphorus, but from carbon and silicon; so that it may be rapidly converted into steel. Iron and steel scrap are not trustworthy as to quality, and they are often dear. There are three methods of purifying cheap materials for the open hearth. 1. Mechanical puddling, as done at Creusot, which removes 90 per cent. of the phosphorus from the pig iron. 2. Krupp's washing process (the conduct and results of which I fully described in a former paper*), which eliminates 70 to 80 per cent. of the phosphorus and most of the sulphur and silicon from pig iron. Neither of these processes would sufficiently purify, for very fine steel, those very impure pigs which are cheapest in many parts of the United States. 3. The process of producing directly from the ore an iron which is practically pure chemically, although mechanically mixed with the impurities of the ore. This is the oldest of iron processes; one form of it, the Catalan forge, employed to produce charcoal blooms, is still in use, but its great cost is rapidly throwing it out of competition.

Among the modern attempts to produce iron direct from the ore, on a large scale and at a cheap rate, several have been in various respects successful. Dr. Siemens's process of treating a ton and a half or more of ore and the coal to deoxidize it, in a rotating gas furnace, and bringing out, in some four hours, a ball of chemically pure iron so soft that the fluid and impure slag may be squeezed out of it, is the most attractive and the most highly developed of all the modern direct processes. I have watched it, from time to time, since 1874, and have noted a steady improvement. It may now be said to have passed the experimental stage, although, like older processes, it must be adapted by practice to special materials. The cause which has been more potent than all others, including the defects of the best direct processes, to bring these processes into disrepute, is the wasteful treatment of the direct *product.* With one or two exceptions (when failure was due to other and obvious causes), the direct product has been made into wrought iron (weld iron). Even Dr. Siemens has, at Towcester, in England, and at Park, Bro. & Co.'s, in Pittsburg, set up his apparatus for this purpose. Is it likely that

* This volume, p. 156.

a commercial success would follow such conditions as these? Here is a red-hot ball of chemically pure iron, mixed with the unreduced refractory sand and clay of the ore. It is not so soft that all the dirt can be squeezed out, so it must be reheated till the materials it incloses are nearly melted. Then this pure iron mass, white-hot for oxygen, is pulled out into the open air, slowly hammered, piled, reheated and rehammered, till about half of it is changed to ore again. If, on the contrary, this ball of direct metal is simply squeezed to expel the bulk of the already fluid slag, which contains most of the phosphorus, and then quickly put under the bath in the open-hearth furnace, no more oxidation can occur. The iron being already hot, quickly dissolves, and the dirt being released, floats on the surface by difference of gravity.

Hundreds of tons of direct metal made at Towcester have been sent to the open-hearth works at Landore, where it quickly melted in the open-hearth bath, and made excellent steel, although the ores from which it was made contained about 2 per cent. of phosphorus.

The apparatus or "rotator" (illustrated on the accompanying plate) consists of a revolving furnace, like a Danks furnace, lined with oxide of iron. Gas from producers and air from one pair of regenerators enter at one end of the furnace, burn and reverberate within it, and pass out at the same end into the other regenerator. There is a large charging and discharging door at the other end of the furnace.

At Dr. Siemens's works at Towcester, the small rotator, 9½ feet long by 8½ feet in diameter, takes a charge of 30 cwts. of ore mixed with 8 cwts. of small coal. In about 2½ hours the reduction of the ore is completed; the slag is tapped off, and the heat and speed of rotation are increased to form the mass into an elongated ball, which is hammered into a bloom. An average of forty-three consecutive charges at Towcester gave the following results:

Iron in ore charged, pounds,	1274
Coal, pounds,	728
Time for operation,	3 hours, 12 minutes.
Blooms made, pounds,	1113
Loss, per cent.,	12.6
Coal in producers per ton of blooms, tons,	2

The particles of iron forming the blooms, if perfectly separated from the slag, are practically pure, however impure the ore may be. The slag contains sometimes 6 per cent. of phosphoric acid and 1 to 2 per cent. of sulphur. The pure iron will alone remain in the open-hearth bath, although some few hundredths of phosphorus may be

taken up from the slag at the highest temperature. The bars hammered from the direct Towcester ore blooms contained (eight analyses): Maximum phosphorus, 0.08; minimum phosphorus, 0.019. The phosphorus in three blooms was 0.019, 0.046, 0.083, while the phosphorus in the ore averaged 2 per cent.

The first trial of the process in the United States was at Park, Bro. & Co.'s works, in Pittsburgh, two years ago. There were no serious difficulties, except the oxidation referred to, in the manufacture of the balls into wrought iron. I am informed that this company intend to start the rotator again to make material for their new open-hearth furnaces. Within the last few months a large rotator, 11 feet long by 11 feet in diameter, has been started at Tyrone Forges, Pennsylvania, by Mr. Robert J. Anderson, of Pittsburgh, to make material for his open-hearth furnaces. Although the operations have purposely been experimental, with various ores and lining materials, enough has been done to show that a product of excellent quality may be got from any ore, and that linings (necessarily oxide) may be adapted to any ore, although a very siliceous ore requires the use of so much lime that the repairs of linings are proportionately increased.

In an average week's work at Tyrone, with Robinson ore and the highly siliceous Pennington ore, the mixture having about 50 per cent. of iron, the charges were: Ore, 4000 pounds; reducing coal, 600 to 700 pounds; limestone, 250 pounds; scale and cinder, 800 pounds. The yield of blooms was 1600 to 1700 pounds per charge, or 80 to 85 per cent. of the iron in the ore. The producer coal was 3800 pounds per ton of blooms. The week's work was nineteen operations, producing 14 tons of blooms.

The cost of blooms, with ore averaging about $3 and coal $2.15, and with labor charged at the very high rate of $10 per ton, was a little over $25 per ton. Experimental labor is of course excessive, and in this case the men could have just as well run four furnaces as one. Labor should not exceed $2.50 to $3 per ton in a plant of four rotators. The output has been gradually increasing, and has reached five operations per twenty-four hours. The producer coal has also been gradually decreased. Of course, working costs can be only approximately determined from experimental costs, but it seems safe to say that blooms can be produced at a small advance over the cost of pig from the same ore.

The cost of a plant of four rotators, ore-crushers, hammer or squeezer, etc., exclusive of building, is about $40,000, and its output, with existing appliances only, in regular rather than in experimental

work, is estimated at 125 tons per week. This looks at first like a small output, but it must be remembered that the entire blast-furnace plant is dispensed with. An obvious improvement, not in any way experimental, is about to be introduced. It is calcining the ores in any suitable kiln, and running them red hot into the rotator. As about half the time of the operation is now occupied in getting the charge up to a reducing temperature, it is obvious that the calcining —a cheap operation—will nearly double the output of a rotator plant.

Charcoal blooms are at present the best material in the market for making fine open-hearth steel; they are used together with the smallest possible bath of Bessemer pig for the finest fire-box plates. If Siemens direct blooms (even should they have more mechanical impurities) are not as good as charcoal blooms for open-hearth steel, the reason is not obvious. Such practice as there is seems to prove them equally good.

As I have similarly stated in previous papers describing new processes, the object of these notes on the Siemens process is not to compare it commercially with other preparatory processes, but simply to state its existing *status* and the probable course and means of its further development.

THE HEAT OF THE COMSTOCK LODE.

BY JOHN A. CHURCH, E.M., PH.D., NEW YORK CITY.

IN May, 1878, I had the honor of presenting to the Institute, at the Chattanooga meeting, some observations upon the heat of the Comstock Lode, and since then the subject has attracted some attention and criticism, in the course of which, I have observed errors of apprehension which I beg leave to correct now. Mr. John Arthur Phillips discussed it before the Geological Society, his paper being printed in the *Quarterly Journal* for August, 1879.*

His criticism is based upon that part of my paper in which I quote a number of analyses of the mine waters, to illustrate the fact that the heat does not come from the oxidation of pyrite. I pointed out that these waters contain only an insignificant proportion of sulphuric acid, and not enough to account for even one per cent. of

* A Contribution to the History of Mineral Veins. By J. Arthur Phillips, Esq., F.G.S. Quarterly Journal of the Geological Society for August, 1879.

the heat observed, the calculation being based upon the oxidation of the quantity of pyrite which corresponds to the amount of sulphur in the water. Mr. Phillips endeavors to disprove my theory of kaolinization, by applying the same reasoning and calculations to the amount of alkalies and alkaline earths dissolved in the waters. He says:

"The average proportion of alkalies contained in the rocks of the district is 6.40 per cent., while the mean of the published analyses gives 11.30 grains of alkalies in 58,373 grains of mine water. It consequently follows that the 4,200,000 tons of water annually pumped out of the workings must contain 813 tons of alkalies, and that, as these are present in the rocks in the proportion of 6.40 per cent., the felspar in 12,703 tons of rock must be annually kaolinized and the whole of the felspars removed in solution.

"The amount of rock in which the felspar has been kaolinized being 12,703 tons and the number of tons of water 4,200,000, it follows that $\frac{4.200,000}{12,703} = 330$ is the number of tons of water heated by each ton of altered rock.

"In order, therefore, that one ton of rock should be enabled to heat 330 tons of water only 1° Fahr., and if the specific heat of these rocks be taken at .1477, which is that of blast-furnace slags, it would require to be heated by the kaolinization of its felspar to a temperature above that of molten gold. Consequently to raise the water 85° or to a temperature of 135°, at which it issues, the kaolinization of the felspar in each ton of rock would require to elevate it to an extent we are unable to estimate, since there is no means of ascertaining the specific heat of bodies at such enormously high temperatures."

Mr. Phillips therefore concludes "that the kaolinization of felspar is no more than the oxidation of pyrites, an adequate cause to account for the heat of the Comstock Lode." His criticism is, however, founded on fallacious premises. It may be possible to use the analyses of mine waters to determine the function of pyrite in the physics of the lode, because the products of the oxidation of pyrite are completely soluble, and all of the sulphuric acid produced may be expected to appear in the water, except such part as is precipitated as gypsum by the lime of the rocks. Observation showed that the quantity of sulphur removed by this precipitation is very small, and my use of the water analyses to exhibit the inadequacy of pyritic oxidation as a source of heat was, perhaps, pardonable, though it was

inaccurate, for the reason that no attempt was made to compute the quantity of heat developed by the *hydration* of the sulphuric anhydride. This omission was allowed because the argument against pyritic oxidation as a principal source of the heat did not depend on heat calculations, but on the fact which I repeatedly stated, that the rocks do not contain pyrite or precipitated ferric oxide in quantity sufficient to account for the heat by oxidation. The calculation based on the analyses merely gave a *measure* of this inadequacy, and showed that a cause to which 100 per cent. of the heat had been ascribed, really did not account for as much as 1 per cent. of it.

But the case is very different with kaolinization. Here we are not dealing with the *least* apparent cause of heat, but with that one which observation shows to be the most extensive result of chemical action in the lode rocks. I believe my theory of heat from kaolinization is acknowledged on all sides to be well founded as a general fact, and the only difference of opinion is upon the quantity of heat obtainable by the hydration of aluminic silicate. My critics acknowledge that where kaolinization takes place some heat is produced, but they deny that its effects can be so remarkable as to produce the temperatures found in the Comstock.

The point which I now wish to insist upon is that pyritic oxidation and kaolinization do not stand upon the same footing as possible sources of heat in the Comstock. Both come under the practical observation of the geologist, and while one is so limited in its quantity as to be insignificant as a factor of heat, the other is found to exist in the excessive magnitude which the student of geology sometimes encounters.

The Comstock rocks may be said to form a mass four miles long by four miles wide, and say two and a half miles thick, and so far as observation teaches us, it shows that every pound of this block has undergone alteration to some extent. A large proportion, say ten per cent., as a *minimum*, has been altered to flaking clay, while a still larger proportion has advanced so far in decomposition that alteration sets in vigorously when a gallery is opened in it, the rock "slakes" and the air of the gallery is highly heated, its atmosphere often becoming temporarily insupportable by man.

It is a mistake to suppose that the results of this extensive kaolinization are represented in the mine waters. It is probable that much of it is accomplished by aqueous vapor, which is entirely absorbed by the rock and does not give rise to any solution. The

rock is not saturated with water but, on the contrary, analyses show that the hydration is almost everywhere incomplete.

The error in Mr. Phillips's calculation is contained in the words, "as these (the alkalies) are present in the rocks in the proportion of 6.40 per cent., the felspar in 12,703 tons of rock must be annually kaolinized and the whole of the alkalies removed in solution."

This conclusion is based upon the supposition that the result of kaolinization at a great depth is clay free from alkalies. But that is not the case. The result is merely altered propylite, or a decomposition product which differs from the original rock only in the addition of water and the removal of a small part of the alkalies. The difficulty, and perhaps impossibility, of obtaining rock which has not been altered in the Comstock region, makes it a delicate matter to institute comparisons between the clay and the original rock, but the following analyses given by Mr. Clarence King* will serve to show the change produced. The composition given for the clay is the mean of four analyses of clays taken from different points, and probably resulting from the alteration of two rocks, propylite and andesite. Analyses of both of these rocks are adjoined, but the specimens did not come from the same places, though they are from the same region. They were chosen out of a series because they are the only examples which are nearly free from water.

	Clay.	Propylite.	Andesite.
Silica,	56 24	64.62	58 33
Alumina,	15 28	11.70	18.17
Oxide of iron,	4.00	8.39	6.03
Lime,	4 40	8.96	6.19
Magnesia,	3.01	1.18	2.40
Soda,	.95	3.13	3.20
Potassa,	3.24	1.95	3.02
Water,	6.26	1.02	.76
Carbonic acid,	2.34	. .	2.85
Phosphoric acid,	.08	. .	. .
Pyrite,	4.36	. .	. .
	100.16	100 95	100.95

In order to obtain an analysis of nearly unaltered propylite it was necessary to take one which does not contain the usual proportion of alumina, while the alkalies are in corresponding excess over the proportion usual in the immediate Comstock grounds, so that the comparison is not entirely accurate. Still the analysis of the clay is

* United States Geological Expedition of the Fortieth Parallel, vol. iii, p. 89, and vol. i, p. 560 and 576.

the main thing, and the mean composition given shows what is invariably the fact in this locality, that the clay retains a large proportion of iron, alkalies, and alkaline earths. It is impossible to say whether more or less than one-half of the soluble products have been leached out, and therefore it is incorrect to say that the solids present in the mine waters represent the kaolinization of 12,703 tons of rock, as Mr. Phillips assumes. The true quantity of *leached* rock may be many times as great as this even in wet ground.

But the quantity of rock that yields solids to the mine waters by no means represents the quantity that is kaolinized. These waters do not permeate the whole mass of rock. On the contrary, nine-tenths of it is extremely dry, and the water is confined to narrow courses, which discharge an unceasing flood from a few points. The dry rock seems to be as hot as the rest, and, like the clay, its analysis exhibits the presence of water. One analysis made in a specimen that showed no sign of alteration, and was classed as a typical specimen of propylite, was found to contain 6.53 per cent. of water, or more than the mean given above for the clay. It is quite possible that the alkalies in the mine waters do not represent one hundredth part of the actual kaolinization.

I have shown that this rock is permeated by currents of gas, which is frequently discharged from the drill-holes with force sufficient to move the flame of a candle. Probably this gas is partly composed of aqueous vapor, which is not perceptible to the eye in the hot levels of the mines, but is seen as a rushing column of steam forty feet high at the mouth of the shaft, where it encounters the atmospheric temperature. All the upcast shafts of the lode show this column of watery vapor, and it appeared to be just as strong in the Imperial, Bullion, and other mines, where fifty to one hundred men were employed, as in the California and Consolidated Virginia mines, each of which gave work to six or seven hundred men. The quantity of steam in the upcast is much greater in the Comstock than in any other mines I have ever seen, though they are by no means the most heavily manned, nor the most watery excavations that have come under my observation.

It is extremely probable that even the dry rock is supplying heat by kaolinization through the action of water vapor over large areas where it never receives water enough to be robbed of its alkali by solution, or represented in the mine waters.

For these reasons the conception of Mr. Phillips, that the rock "would require to be heated by the kaolinization of its felspar to a

temperature above that of molten gold," has no foundation either in the theory or facts presented by me. The problem laid before us in the mines of the Comstock is as complex as it is interesting. I am justified in saying of all other theories that they do not even attempt to explain the facts. The explanation I give is still merely a hypothesis, but at the present day it is unique in being based upon a careful statement of the observed phenomena.

I have explained in my previous paper, and also in a subsequent publication,* that the movement of gaseous currents in the solid rock has the effect of bringing each part of the mass under the influence of all the heat produced in the rock below it, for the gas is one of the results of kaolinization. It is released at the moment of alteration, and has no alternative but to take its way to the surface, carrying with it part of the heat produced, which it distributes through the whole mass of rock.

It is probably owing to the operation of this gaseous current that the immense quantity of heat observed is discharged into the mine levels, and the water may receive part of its heat in the same way. A water-course and a mining gallery are both localized things placed in fixed positions in the pathway of a hot gaseous current, and the heat they receive may be (and in the case of a gallery with the air in it certainly is) derived in part from alteration proceeding in the rock at considerable distances from them.

Another of my critics is Prof. G. F. Barker, of the University of Pennsylvania, who is reported to have said that the rock of the Comstock is not uniformly heated to 130° F., the remark being made to Dr. J. P. Lesley, and apparently as a contradiction of a statement made in my paper. Professor Barker's remark, as quoted by Dr. Lesley, was "that there was no uniform temperature; but, on the contrary, the most remarkable differences, some of the higher levels being much hotter than some of the lower levels." This state of things was fully exhibited by me; and I showed that, while the great mass of rock near the two-thousand level, had a "pretty uniform temperature of 130° F.," it also contained narrow belts, most of which are hotter, while some are colder than the general mass. In any given mine the drift on a high level may be run for a long distance in a hot belt, and be much hotter than a drift two hundred feet lower, if the latter is carried through what I have called the "general mass" of rock. I regarded the extreme local temperatures

* The Comstock Lode, its Formation and History, published by Wiley & Sons. New York, 1879.

as significant phenomena, the explanation of which would carry with it the whole theory of heat production, and discussed them to that end.

Professor Barker's opinion is, that the heat is a hot-water heat, and that the water is heated by movement of the rocks. It is true that incessant and great movement of portions of the rock is encountered in the mines; but the opinion in question does not take into consideration the character of this movement. Every miner on the lode will sustain me in saying that the motion consists in the swelling of parts of the rock when its conditions have been artificially altered by the excavations of the mines. The usual explanation is, that this swelling is caused by the admission of air and moisture brought in by the artificial openings, an impression in which I fully concur. There are no indications that this movement takes place at a distance from the drifts, and there is no sign of natural movement at the depth of two thousand feet sufficient to produce as much heat in a year as the complete combustion of 28,601 tons of carbon, this being a low estimate of the quantity of heat withdrawn in the mine waters and air.

In addition to the doubt whether extensive and continuous movement of the rocks does exist, this theory of heat production fails to explain how the immense quantity of heat is carried to the limited localities where the water and air are obtained. Mere conduction through the rock does not suffice unless co-efficients other than those in ordinary use are employed.

In this connection there is great significance in the fact that the first thousand feet of depth does not exhibit unusual increase of temperature, though this is especially the zone of oxidation, solution, and consequently of movement resulting from changes of volume.

I am indebted to Mr. Charles Forman, of Virginia City, Nevada, for valuable facts upon this subject. He is superintending the sinking of a shaft which is designed to reach the lode at a depth of about four thousand five hundred feet, and, with enlightened regard for exact knowledge on this interesting subject, he procured for use in the shaft a Negretti & Zambra slow-action thermometer, of the pattern adopted by the Underground Temperature Committee of the British Association, and standardized at Kew. The shaft is now one thousand feet deep, and I have received from him the following data obtained by the use of this instrument:

"Temperature of the ground in Forman shaft, from the surface to the depth of one thousand feet, as ascertained by drilling holes

three feet deep into the rock and inserting a Negretti & Zambra mining thermometer in the hole; closing the hole with clay, and leaving the thermometer for twelve hours; not less than three holes being tried at each point:

Depth.	Temperature.	Increment.	Decrement.
100 feet,	$50\frac{1}{2}°$ F.		
200 "	55°	$4\frac{1}{2}°$	
300 "	62°	7°	
400 "	60°		2°
500 "	68°	8°	
600 "	$71\frac{1}{2}°$	$3\frac{1}{2}°$	
700 "	$74\frac{3}{4}°$	$3\frac{1}{4}°$	
800 "	$76\frac{1}{2}°$	$1\frac{3}{4}°$	
900 "	78°	$1\frac{1}{2}°$	
1000 "	$81\frac{1}{2}°$	$3\frac{1}{2}°$	

The increase of temperature is $31\frac{1}{2}°$ in nine hundred feet, or precisely $3\frac{1}{2}°$ in one hundred feet. This determination has especial value, because the shaft is nearly a mile and a half east of the lode, and therefore not controlled by its physical conditions. The shaft has been quite wet, and its temperature may have been affected by the presence of water. This series of observations promises to be extremely valuable.

I have discussed this subject at length because of the interest which has been shown in it and because of its intrinsic importance. The Comstock Lode is unique as *the* hot lode of the world. I have followed up several reports of hot veins elsewhere and have learned that while "hot springs" are known in other mines, there is no other metalliferous deposit of which I can hear that presents the peculiarity of this locality which may be stated as *hot rock*, comprising the whole country of the lode. The Mexican mines are hot on account of bad ventilation, but I have ascertained that the rock is not hot. The quicksilver and sulphur deposits of California have shown great heat in a superficial stratum, but I am informed that the heat disappeared at the depth of a few feet. Another mine near the Comstock was reported to be hot, but this turned out to be heated by a steam-pipe which the geological observer had not noticed. At Steamboat Springs, seven miles northwest of the Comstock, there is a mass of quartz which has been mined for a small proportion of mercury which it contains. This is really another hot locality, and its heat is the more remarkable because the mass is almost pure quartz. Half a mile away are the Springs which steadily pour out a current of boiling water, from which silica, with occasional traces

of sulphur, cinnabar, and even gold, are said to be deposited. Probably the heat of the quartz body is intimately connected with the source from which the Springs derive their heat, and the whole locality is in the Comstock neighborhood.

Mr. R. Pearce, of Argo, Colorado, has kindly called my attention to the analysis of a hot spring in a Cornish mine, which he investigated several years ago. The analysis was made by the late W. A. Miller, M.D., Treasurer of the Royal Society and Professor of Chemistry in King's College, London. It was published in the *Chemical News*, October 15th, 1864, p. 181, and is given below. It presents two remarkable peculiarities, showing in one gallon no less than 216.17 grains of calcic chloride, and 26.05 grains of lithic chloride. The water is described as forming "a spring" in the mine.

ANALYSIS OF CORNISH MINE WATER.

From a hot spring in Wheal Clifford Mine, Cornwall: Depth, 1320 feet below mean level of the sea. Flow, 150 gallons per minute. Temperature of water 125° F. Sample collected by Mr. R. Pearce. Analysis by Prof. W. A. Miller, F.R.S.

Specific gravity at 60° F.,	1.007		
Gases—			
1 imp. gallon, at 30 in. bar. and 60° F., contains,	8.91	cubic inches.	
Consisting of—			
Carbonic acid,	1.89	"	"
Ratio of O. to N., 1 : 3 { Oxygen,	1.72	"	"
Ratio of O. to N., 1 : 3 { Nitrogen,	5 30	"	"
Solids—			
1 imp. gallon contains fixed salts, by evaporation,	646.10	grains.	
Consisting of—			
Chloride of lithium,	26.05	"	
Chloride of potassium, with a little chloride of cæsium,	14.84	"	
Chloride of sodium,	363.61	"	
Chloride of magnesium,	8.86	"	
Chloride of calcium,	216.17	"	
Sulphate of calcium,	12.27	"	
Silica,	3.65	"	
Oxides of iron, manganese, and alumina,	traces.		
	645.45	"	

THE NORTH STAFFORDSHIRE COAL AND IRON DISTRICT.

BY WM. HAMILTON MERRITT, F.G.S., ST. CATHARINES, CANADA.

In this paper, which I have the honor to submit to the Institute, it is my intention to treat especially of that part of the North Staffordshire field which converges to a long tongue in the neighborhood of Congleton, and which includes the villages of New Chapel, Ford Green, Norton, Biddulph, Bradley Green, Gillow Heath, and the immediate vicinity.

My object is not so much to give a minute description of the geology or systems of mining and smelting, as by a general review, some idea of the extraordinary facilities for the production of excellent iron, which have enabled this district to maintain its position more successfully than any of its rivals against the late depression in trade.

The North Staffordshire coal field has the Cheshire and Lancashire fields some thirty-five miles to the north, those of Derbyshire and Nottinghamshire forty miles to the east, those of South Staffordshire and Shropshire, about thirty miles to the south, and the Denbighshire and Flintshire fields some forty miles to the west. It is highly probable that the coal continues under the new red sandstone to the Western and Southern fields as the dips on both sides, and absence of large faults, make it impossible to come to any other conclusion. In a section of this coal field, from Chatterly to Whitfield, thirty-two workable seams of coal are shown, of an aggregate thickness of 130 feet, varying from 2 feet 6 inches to 7 feet, and thirteen seams of ironstone, 24 feet, averaging from 2 to 4 feet. All of these seams have been minutely described in a paper by Mr. Charles J. Homer, read in 1875 before the British Iron and Steel Institute. In the northern part, to which I especially wish to direct your attention, the beds lie in a V-shaped basin, the underlying millstone grit and Yoredale rocks rising and forming escarpments on either side of the valley containing the coal.

Throughout this district, at the outcrop of the coal, there are indications of old workings. A shaft 4 to 5 feet in diameter reaches the coal at about 14 yards from the surface, and from it the coal has

been worked away by a mode which appears to have been similar to the "punch and thirl" system. The greatest depth of these old workings is generally not more than 16 yards, and they are drained by an adit from the lowest point. In one of these an oaken shovel was found buried in the *débris*, and as the workings did not seem to date back further than the latter part of the sixteenth century, good mining work has evidently been successfully accomplished (the shaft of this pit being of admirable construction) with far ruder implements, in many cases, than we are aware of.

Owing to the high dip of the beds of this coal field, 14 to 16 inches to the foot, the coal has to be worked by a system of pillars 40 feet long by only 10 feet thick, the great inclination not allowing them to be any broader. The mode of working adopted throughout this section can be explained in a few words. The main level, or "horse-road," is driven horizontally with the cleat of the coal, together with an "air-head" 10 yards off on the upper side, which is "thirled" into every 20 yards for ventilation. This air-head afterwards takes the return air from the workings to the "up-take," and no miner is allowed to enter it. At every 160 yards a "bord" or "brake-dip" (so called from a brake situated at the top to allow the full car to pull up the empty one) is driven "up bank" at right angles to the main level, the longest being 110 yards. A smaller bord for air is likewise run up beside this and thirled into every 10 yards. On reaching the top, two drifts are put out from each brake-dip at 10 yards apart, and the air "bratticed" up by canvas for 40 yards, when it is thirled. These drifts from either side meet at 80 yards, and after thirling again, the narrow pillars (of 40 by 10 yards, as before mentioned) are worked away against the cleat, the air being made to pass along the face of the work, and then at once to the up-take air-head. As soon as taking away the pillars above is fairly commenced the next drift is driven, so that the working of the pillar above is always slightly in advance of that immediately below.

The ironstone in this district occurs, as before mentioned, in beds of about 2 to 4 feet in thickness, interstratified with the coal, consisting chiefly of solid bands, with occasional nodules, and sometimes containing shells of the bivalve *anthracomya*. As they are worked nearer the surface than the coal on the west side of the valley (well seen at New Chapel), the dip is inconsiderable. Owing to the thin beds, ponies are used entirely, and the mode of getting the stone to the main levels is one of the most primitive to be seen anywhere.

A man loads the ore on a wooden sleigh, which he then drags, on his hands and knees, through a small road made through the gob by packing up with stones on each side. The ironstone is worked away in a face of 30 yards, by "holing" the shale below and putting the shot in above.

The facilities for iron-smelting will be readily conceived when it is taken into consideration that we have blast furnaces situated at the mouths of pits, which work the seams of coal mentioned above, the ore a short mile off and beds of carboniferous limestone close at hand, worked by the Astbury Lime Company. The advantages of this district will be the better realized, however, if I give analyses of some of the coal and ironstone. The analysis of the harder sort of coal, which contains little or no sulphur, and which works admirably in the furnace, I am unable to give; but of samples richer in hydrocarbons from the same district, the two following are good examples:

	Bucknall.	Whitfield.
Carbon,	65.69	67 60
Hydrocarbons,	31.52	31.30
Sulphur,	0.04	trace
Ash,	2.75	1.10
Total,	100.00	100.00

Cokes from above:

	Bucknall.	Whitfield
Carbon,	95.82	98.38
Sulphur,	0.00	0.00
Ash,	4.18	1.62
Total,	100.00	100 00

The ironstone is calcined in heaps at the mine mouth with inferior coal. Before the operation it varies in richness from 34 to 58 per cent, of the protoxide of iron, and afterwards runs as high as 90 odd per cent. of the peroxide. The following analyses of the "Red Shag" and "Red Mine" ores give a very just example of the ironstones most in use:

	Red Shag.		Red Mine.	
	Raw.	Cal'd.	Raw.	Cal'd.
Peroxide of iron	....	91.50	....	92.52
Protoxide of iron	45.30	....	50.90	....
Peroxide of manganese	....	4.12	....	2.44
Protoxide of manganese	2.23	....	1.76	....
Alumina	0.32	0.55	0.73	0.59
Silica	0.50	0.86	1 13	0.79
Lime	0.64	1.10	2.07	1.79
Magnesia	0.20	0.34	0.80	0.94
Sulphide of iron	0.32	....	0.10	....
Phosphoric acid	0.90	1.53	0.62	0.93
Combined water	1.07	....	0.12	....
Carbon	18.60	....	8.75	....
Carbonic acid	29.92	....	33.02	....
Total	100.00	100.00	100.00	100.00
Metallic iron	35.40	64 05	39.58	64.77
Metallic manganese	1.72	2 97	1.35	1.75
Total	37.12	67.02	40.93	66.52

Owing to the excellency of its raw materials, this district produces iron unsurpassed by any in the English market, and its boiler plate more than successfully rivals the produce of John Brown & Co., of Sheffield. A statement appeared in the *Journal of the Iron and Steel Institute* for 1875, "that the iron manufactured in North Staffordshire with pure coal was realizing a higher price than any iron in the market."

One of its greatest producers, Robert Heath, M.P., Vice-President of the Iron and Steel Institute, furnished the iron for the greatest work of its day, namely, the Victoria Bridge of Montreal. It is chiefly owing to this gentleman's industry that North Staffordshire occupies the position she now does among the iron districts of the country, and his enterprise has been rewarded in that he is the largest private ironmaster in Great Britain.

The average height of the furnaces in this district is from 50 to 70 feet; those using coal will not stand a greater height than the latter, to which most of the new ones are being raised. As the coal causes a little caking in the hearth, a process is in some places resorted to which is seldom met with elsewhere. The furnace is found to work better if the sides and bottom of the hearth are cleaned once in every turn of 12 hours, and this is done by running in long bars to loosen the crust, which is then blown out with the steam generated from a bar dipped in water before it is thrust down.

An average charge is as follows, No. 1 being the quantities used

for Red Shag alone, and No. 2 for a mixture of Red Mine (raw 50 per cent.), and Lean Mine (raw 35 per cent.).

No. 1.	Cwt.	Qrs.
Coal,	29	0
Ore,	29	2
Limestone,	2	2
Flue cinder,	1	3

No. 2.	Cwt.	Qrs.
Coal,	22	2
Ore { Red Mine,	19	1
Ore { Lean Mine,	16	2
Limestone,	5	0
Flue cinder,	2	1

The consumption per ton of pig is about:

	Tons.	Cwt.	Qrs.
Coal,	1	15	3
Ironstone,	1	14	4
Limestone,	—	8	2
Flue cinder,	—	4	2

The puddling is altogether manual, a number of Danks's puddlers in this neighborhood, about the first erected in the country, having been stopped at the commencement of the late depression in trade.

With regard to the cost of production, I might state that in 1877 the coal could be mined in this district at a little under $1 a ton, and the ironstone from about 75 cents to $1.15 a ton. The work of getting is let out at so much per ton (averaging in that year for coal from 50 to 75 cents per ton), the contractor employing miners whose daily gain averaged about $1. In the same manner, by taking the immediate wages at the blast furnace into consideration, a ton of pig iron could be produced at a very little over $1 a ton for labor. It will be remembered that wages have gone down since 1877, and if they have not yet reached the old figure, these approximate prices would have to be reduced still more to arrive at the present cost of production in this district.

I have briefly placed these facts before the Institute to show, by a good typical example, why England produces iron at a price, with which, at present, on equal terms, it is almost impossible to compete.

THE MINERAL RESOURCES OF SOUTHWESTERN VIRGINIA.

BY C. R. BOYD, WYTHEVILLE, VIRGINIA.

The region to which I have the pleasure of calling your attention, though limited in area, is remarkable for the quantity and purity of its mineral deposits, and in these respects it would be difficult to find its equal anywhere.

IRON ORES.

The red and brown hematites, pipe ore, and semi-magnetites, from which is made in charcoal furnaces the highest quality of iron for car-wheels, extend through the counties of Giles, Montgomery, Pulaski, Wythe, Smyth, Washington, Bland, Tazewell, Russell, Scott, Lee, Floyd, Carroll, and Grayson, Virginia, and run over into Ashe County, North Carolina. One locality of semi-magnetite, in the centre of the great Giles County basin, has in sight, by actual measurement, 50,000 tons of ore, which, according to Prof. Fesquet, contains 69.74 per cent. of iron and no phosphorus. In the great brown-ore belt, which passes through the counties of Montgomery, Pulaski, Wythe, Smyth, and Washington, there is an extraordinary deposit of more than a million tons. A small section of this very long vein, on Cripple Creek, in the County of Wythe, yields an ore which, analyzed by Mr. James Aumann, gave the following results:

Metallic iron,	58.15
Water,	12 96
Alumina,	2.32
Silica,	1.09
Phosphorus,	none.

At one point towards the western end of Red Land Mountain, in Pulaski County, New River section, I measured a body of ore in this belt which will yield, to a depth of 150 feet, over 8,000,000 tons, and the deposit extends far below the 150 feet measured at the upper part. Again, at Rich Hill, near the mouth of Reed Island Creek, these veins show extraordinary surface development, giving an ore of very great purity. At numerous places, also, in the counties of Giles, Bland, Tazewell, Russell, Scott, Wise, and Lee, these brown and red iron ores are exposed in such vast quantities as to baffle both description and measurement. That of Chestnut Flat, in Giles County, back or west of the Angel's Rest Mountain, is an easily reducible ore, blood-red when crushed, of which there are fully 300,000 tons in sight. Its analysis is as follows:

Sesquioxide of iron,	89 55
Oxide of manganese,	0.20
Silica,	2.58
Alumina,	1.11
Lime,	0.20
Magnesia,	0.15
Sulphuric acid,	0.37
Phosphoric acid,	0.30
Water, hygroscopic,	1 25
Water, combined,	4 10

Among other known localities may be mentioned Round Mountain, Nye's Cove, Newberry's, and other places in Bland County; Whitely's Ridge, Kent's Ridge, and numerous other places in Tazewell County; Kent's Ridge, Copper Ridge, Clinch River, in Russell County; Copper Ridge, Moccasin Ridge, Big Ridge, Newman's Ridge, Powell's Mountain, Boatwright's, in Scott County; the neighborhood of Big Stone Gap, in Wise County; Bales's or Bowling Green Forge, Waldin's Ridge, Poor Valley Ridge, in Lee County.

The strictly fossil ores are found in continuous veins in Walker's Mountain, Gap Mountain, Clinch Mountain, Round Mountain, Wolf Creek Mountain, Pearis's Mountain, Buckhorn, East River Mountain, Peters's Mountain, Paint Lick Mountain, Salt Pond, Butte, Newman's Ridge, Powell's Mountain, Waldin's Ridge, and Poor Valley Ridge, in the counties of Giles, Montgomery, Bland, Pulaski, Russell, Scott, Wise, and Lee. Of all these localities, the best fossil ores I have seen came from the Clinch Mountain, in the line between Washington and Russell counties; Poor Valley Ridge, near Pennington's Gap, in Lee County; Boon's Path, same county; East River Mountain, Giles County, and one or two points in Wolf Creek Mountain, Pearis's Mountain, and Round Mountain, in Bland County.

The Giles and Bland fossil ores assayed, according to H. Dickinson, of Norwood, Mass., as follows:

Sesquioxide of iron,	58.12
Oxide of manganese,	0.06
Alumina,	4.07
Lime,	0.20
Magnesia,	0 41
Potassa and soda,	0.40
Silica,	32.74
Sulphuric acid,	0.00
Phosphoric acid,	0.75
Water, hygroscopic,	0.60
Water, combined,	0.96
Organic matter,	0.84

Other ores from the fossil belt in East River Mountain gave Prof. Fesquet 50.36 per cent. of metallic iron.

The true magnetites are principally found in the more ancient rocks (Laurentian) in the counties of Floyd, Carroll, and Grayson, Virginia, and in Ashe County, North Carolina, in veins varying between 3 and 30 feet. One well-defined vein at Ballou's, on New River, in Ashe County, N. C., is 30 feet in thickness by 150 feet elevation above water in the river, the dip varying between 28 and 60 degrees. For a length of 300 feet it is very accessible to the river, and shows 1,800,000 tons of an ore that yields, according to Mr. John Fulton, 0.031 per cent. of phosphorus. Another portion of the vein holds 0.026 per cent. of phosphorus, by the analysis of Mr. F. P. Dewey.

These veins, of which there are three in the locality just named, are almost continuous through Ashe County, North Carolina, and Grayson, Carroll, and Floyd counties, Virginia, and will yield so vast an amount altogether that they can scarcely be ranked as second to any known deposits. Floyd County, at the Toncray Mines, gives fine magnetite, somewhat south of the general direction of the above veins.

The semi-magnetites are found in the counties of Giles, Montgomery, Carroll, Wythe, Smyth, and Washington. Many of them contain, according to the analyses of Messrs. Booth and Garrett:

Metallic iron, ...	55 690
Phosphorus, ...	0 028

Their quantity is not yet so fully determined as the true magnetites and peroxides. They occur usually as semi-magnetic red ores.

THE LEAD AND ZINC ORES.

These ores are confined principally to the counties of Wythe, Pulaski, Montgomery, Smyth, and Washington, and one or two localities in Bland, Russell, and Scott counties. While occurring in large quantities in Pulaski and Montgomery, their greatest development seems, from all explorations to the present, to be in Wythe County, from near Reed Island Creek to the southwest, along New River and up the waters of Cripple Creek. The ores undoubtedly belong in the rocks of No. 2. It is an error to place these measures in the upper part of the Trenton limestones, as I understand some persons have done.

The extraordinary quantities of carbonate, oxide, silicate, and sulphide of zinc, and sulphide of lead, at different points in Wythe County, suggest the idea of a very large continuous deposit. Perhaps the following measures, taken on Painter's Branch, known originally as the Kitchen's, Noble's, and Painter's mines, may not be uninteresting: Dip of measures 30 degrees northwest, measured with the clinometer in a deep shaft, as well as other places. Beginning on the floor or southeast wall of the main measure, we have 144 feet of heavy blend-bearing strata; then 36 feet of dolomite, with occasional spots of zinc and lead; 36 feet iron sulphuret and oxide; 90 feet dolomitic rock, containing large veins and deposits of zinc and lead sulphuret, one of which is 18 feet thick; 180 feet of iron, zinc, and barytes, heavily disseminated in the rock; then toward the northern or hanging wall an indefinite amount of dolomite, more or less charged with barytes. The hill, along the crest of which these measures were taken, is 75 to 100 feet above the water in the small creek flowing near. This series of rocks trends through the country for many miles in a general direction N. 70° E.

At the Wythe lead and zinc mines, on New River, the great pressure apparently exerted from the southeast throughout this whole region of country, in folding the earth's surface, has met with such resistance as to cause a partial fusion of the various strata holding the lead and zinc. Hence the whole body of the rocks is more crystalline in structure, and has less that appearance of stratification which is so apparent at other points. Here the measures from which this ancient lead company has, under one form or another, for more than 100 years taken its ores, are forty feet in thickness between walls of dolomite, with a dip of 70° in the main drift, which is reached with a tunnel 1600 feet in length. Numerous excavations over this hill show other deposits of lead of good dimensions, as well as large bodies of zinc ores of high grade.

The Bertha mine now shows a face of silico-carbonate and oxide of about 180 feet by a depth of 20 feet, with the ore still below. Numerous shafts in this and adjoining hills show the continuity of the beds from which the recently erected smelting works on the Atlantic, Mississippi, and Ohio Railroad, at Martin's Station, derive their ores; and I have the pleasure of exhibiting the first product ever turned out of a zinc furnace in Virginia, which was tapped from the retorts last week. On the headwaters of Walker's Creek, in Bland County, there are lead and zinc ores, as well as in other localities to the northeast and southwest.

COPPER.

The three great lodes of copper which are known to exist in the section under consideration, pass through the counties of Floyd, Carroll, and Grayson, Virginia, and continue on through Alleghany, Ashe, and Watauga counties, North Carolina. There is another—the native lode—confined to Carroll County, Virginia. This lode differs from the rest in being apparently injected from below, and is associated with tremolite and hornblendic trap, having a trend from N.W. to S.E., while the direction of the first-named lodes is N.E. to S.W. The first three are distinguished as the Northern or Iron Lode (having its greatest development in Carroll County), the Middle or Peach Bottom Lode, and the Southern or Ore Knob Lode. The middle lode shows best at the Peach Bottom Mine in Alleghany County, and at Elk Knob, in Watauga County, North Carolina, while the southern has its finest development at Ore Knob, Ashe County, North Carolina, and at Toncray Mine, Floyd County, Virginia. The Ore Knob Mine, when I inspected it in the early part of this winter, was giving sulphuret ores from a vertical vein, 18 feet thick, averaging 25 per cent. of copper, at a depth of about 300 feet from the surface. This ore is being converted into ingot copper at the mine, by the extensive plant of the Ore Knob Copper Company. At the Toncray Mine, on this lode, the better grades of ores have been exposed in a vein 30 feet thick, but dipping S.E. about 45 degrees, and having on its northern wall 4 feet of excellent magnetite.

On the northern lode, at some points in Carroll County, you will find a thickness of fully 150 feet. At one point, where it gives this measure, a shaft sunk into its central portion shows sulphuret ores which will average 5 per cent. of copper for 30 feet in width, while the remainder of the 150 feet on either side gives only 1.70 per cent. of copper. This great lode is marked by extensive beds of gossan, forming most abundant and useful hematite ores. It may be as well to say that throughout this copper belt there are numerous other minerals, both interesting and highly valuable,—specular ores, mica, feldspar, asbestos, and gold and silver, as at Cowles's, Gap Creek, in Ashe County, North Carolina; gold being also found on Bush Creek, and Little River, in Floyd and Montgomery counties, Virginia, and silver in the ores at Peach Bottom Mine, and at the Clifton opening, near Old Town, on the northern lode. To these, also, may be added nickel, cobalt, antimony, and arsenic.

COAL.

Southwestern Virginia holds a very large area of the southern

portion of the great Kanawha Coal Basin proper; the counties of Tazewell, Russell, Scott, Buchanan, Wise, and Lee, participate in it; while in the counties of Montgomery, Pulaski, Wythe, Smyth, and Bland are found the coals which belong more strictly to the protocarboniferous series, and are designated generally as the Upper New River series. The former may be considered as belonging nearer the great Carboniferous, from the regularity and continuity of both the coal and the accompanying rocks. The latter have, also, been found to run for miles through the counties named with surprising regularity; so much so as to baffle those gentlemen who, basing their opinions on the unreliability of corresponding measures in Pennsylvania, ventured to predict the same character for the Virginia beds. Daddow was one of the first to recognize that these measures were of a highly valuable nature; and recent developments have fully proved his conclusions. Witness the operations of the Altoona Coal Company at Martin's, in Pulaski County; of the Blacksburg Company, and numerous others, in Montgomery County; of Colonel Boyd, and Joseph Crockett, in Wythe County, and others, in different localities. It is undeniably true that much of this area has been badly injured by the convulsions to which this part of the earth's crust has been subjected; but there are large areas which have been preserved in almost their original regularity, or, if disturbed, only to the advantage of the miner. These veins vary in thickness, being 22 feet at Altoona, 8 feet in Montgomery, 5 feet in Wythe, and less at other points, generally with a uniform pitch of 30 and 42 degrees in different parts, except where there are well-defined basins, as in Pulaski, and the lower part of Wythe and Montgomery counties. These dips then apply mainly to the outcrops, while the central portions lie nearly flat.

The southeastern portion of the Great Kanawha Basin, which we have just mentioned as being confined to the counties of Tazewell, Russell, Scott, Buchanan, Wise, and Lee, cannot be overestimated as a mineral producing region. I do not exaggerate in the least when I say that these counties hold the most valuable bituminous coal veins, accessible to the miner, and close to the ores which they are intended to reduce, in the country. Within seventy-five miles of vast quantities of the iron, copper, lead, and zinc ores are found horizontal veins of fine bituminous and block coals, measuring 11 feet, 8 feet, 5½ feet, and 4 feet, running for miles reliably through the extended area above mentioned. It is true that a part of the extreme southeastern edge of this area is somewhat broken up by a

double fault; but immediately back of this begin the flat dips and reliable measures above alluded to.

We have, also, in Southwestern Virginia the largest and most valuable virgin forest in the United States, comprising poplar, cherry, walnut, oak, hickory, white pine, hemlock, and locust.

In conclusion I may add, as of interest to iron manufacturers, the following analysis, made by Mr. Dickinson, of a limestone from the Lower Helderberg group at the base of Flat Top Mountain, Dismal Creek, Giles County:

Lime,	49.42
Magnesia,	2.04
Protoxide of iron,	1.53
Oxide of manganese,	0.15
Alumina,	0.48
Silica,	2.94
Sulphuric acid,	0.02
Phosphoric acid,	0.04
Carbonic acid,	42.00
Water,	0.60
Organic matter,	0.78

DISCUSSION.

MR. O. J. HEINRICH: It is with great pleasure that I see the first specimen of metallic zinc produced within the boundaries of Old Virginia from its native ores. As long ago as 1860 I called the attention of the authorities of the State of Virginia, and afterwards those of the Confederate Government, to the desirability of erecting zinc works on the line of the Virginia and Tennessee, now the Atlantic, Mississippi and Ohio Railroad, to utilize the magnificent calamine which could then be picked up on the old surface waste heaps at the Union Lead Mine near Wytheville. After the close of the war I had charge of these works, and made many efforts to induce the members of the company, mostly men of the neighborhood, who owned the mines, to erect zinc works, but without success. I am glad that a younger brother engineer has come forward as a champion of this hitherto neglected but immensely valuable region. It requires enthusiasm to keep up one's spirits when living in a country where the majority of the people are indifferent to the development of the mineral wealth around them.

I can confirm all that our member from Wytheville has said of this region, and could have given a good many additional facts had I known that this subject was coming up. During 1868 zinc ore was mined at the Union Lead Mines, and shipped to Trenton, N. J., and

to the Lehigh Zinc Works at Bethlehem, Pa. It was sold at the mouth of the shaft at $6 a ton, and the average cost of raising it was $3.50 a ton. At this time I was informed that the cost of pumping the water alone at the Friedensville mines amounted to $6 to the ton of ore. About 2500 tons of zinc ore were raised that year. Double this amount might have been produced but for the shortsighted policy of the proprietors, who would not make the necessary outlay for the improvement of the dressing machinery.

The Union Lead Mines have produced since 1838 12,167 tons of pig lead (2000 pounds), charcoal being used for reduction. Before the war the cost of production was 2.4 to 3.1 cents per pound, during the war as high as 12 cents (gold), and since the war up to 1869 5 cents per pound. If these mines had been in the hands of a vigorous company four times this amount of lead could have been produced. As much as 745 tons have been raised in one year, but the necessary exploration being neglected this production could not be maintained. A tunnel 1100 feet in length, which cost $32,000, is now used only as a tram road, connecting with an old shaft which is used as a shot tower, when it might be the main avenue of large and productive mines.

I have surveyed nearly all the old works of this mine, many of which are now inaccessible. It is much to be regretted that proprietors are often so blind to their own interests as to reject the advice of experts. The time may come when the true value of the property will be understood, and the information developed by my surveys is at the service of those who know how to appreciate it.

A good point for the establishment of manufactures of various kinds is on the New River, in the neighborhood of Central Station on the Atlantic, Mississippi and Ohio Railroad. I recommended this to the Confederate Government, when in charge of its explorations in this region, as a desirable site for the proposed National Foundry. Iron ores are here found in great variety and inexhaustible quantity, with abundance of ores of zinc, lead, and copper, and iron pyrites. We have here, too, deposits of salt, limestone, fire-clay, building-stone, and semi-bituminous coal, and the bituminous coals of West Virginia, Kentucky, and Tennessee are all within a limited radius.

Another site for the foundry, even more favorable with regard to iron ores, was recommended in the vicinity of Covington or Buckhannon on the James River. But the vigorous march of General Sherman put an end to work here before the foundation-stones were laid.

I could dwell for hours on the vast mineral resources of the Old Dominion, and I am only too glad that the attention of professional men and capitalists is now being called to this interesting and valuable region.

DR. EGLESTON: The details which Mr. Boyd has given us of the resources of the State of Virginia are, in my opinion, rather under than overstated. I spent two months in this State, during the past summer north of the district which Mr. Boyd describes, examining its resources, and have for two years or more been called upon to examine ores of different kinds brought to me from there.

The iron resources of the western part of the State are especially remarkable, both for the quantity and quality, as well as for the variety of the ores which occur there, and also for the manner in which they occur, which renders their mining so extremely easy. Commencing at the Maryland line the whole belt of country which lies between the Blue Ridge and the Allegheny Mountains may be said to be ferriferous. On the eastern side of the Blue Ridge the Archæan ores are found, which are followed on the western slope by the ores of the Potsdam period, greatly developed, then the Clinton and Oriskany groups. Incidentally deposits are found in other formations, as in the Trenton limestone, or as it is there called the Valley limestone, and also in the Hamilton shales. These last are pockety, while the others are regular geological beds. Independently of these are the magnetic belts of the James River in the southwestern part of the State. It is astonishing that so little attention has been called to the mineral wealth of the State of Virginia. There seems to have been a theory that the Oriskany sandstone, which is the great ore-bearing formation of the State, was too rich in silica to be worked. This finds expression in the *Geologists' Travelling Handbook*, in which it is distinctly asserted that the Oriskany formation contains no ores that can be utilized for the manufacture of iron.

Shortly beyond the Maryland line the great Shenandoah Valley is cut in two, in the direction of its length, by the Massanutten Mountains, and in their valleys the Potsdam ores are worked on the western flanks of the Blue Ridge, while the Clinton and Oriskany ores are worked on both slopes of the Massanutten Mountains. Not far from Staunton these mountains abruptly terminate, and the valley opens for several miles the whole width between the two ranges. Commencing at Buffalo Gap, the ores reappear on the western flanks of Little North Mountain. The Clinton ores out-

crop occasionally, and in some places, as at Clifton Forge, are well developed, in three beds about one foot each in thickness, and have been mined to a considerable extent. The Oriskany is, however, the formation which usually appears in force. It has been folded on itself and afterwards eroded, so that for a distance of more than fifty miles, it crops several times at short intervals high up on the hillside, in beds of from twelve to twenty feet in thickness, and in a few localities even thicker. Very little work is done on the Potsdam beds. In the Massanutten Mountains the most prominent ore is that of the Shenandoah Iron Works, where a small charcoal blast furnace is being worked. The only other furnaces at work, at the present time, are those at Longdale and Quinnimont, though an eighty-ton furnace is being constructed at Low Moor.

The coals of West Virginia are remarkably pure and free from sulphur, and are as low as two per cent. in ash. I have made a large number of analyses of the cokes, and have found them to contain rarely more than six per cent. of ash, which is a much lower average than that of the Connellsville coke. These averages of the coals and ores are given of over one hundred samples, which have been collected by myself and others, which I have very carefully analyzed.

I did not explore the magnetic regions of the James River, and of the southwest part of the State; but the samples which I have seen of the ores from there fully justifies what Mr. Boyd has said of them.

The Virginia iron deposits justify the erection of a very large number of furnaces. Very few of the ores are suitable for making Bessemer pig, but for other purposes the ores are of good quality and rich; the coke is excellent and remarkably free from ash, sulphur, and phosphorus, while good limestone and plenty of water can be had in abundance.

Two years ago my attention was called to the zinc mines in Wythe County. I was totally unprepared to find them entirely free from lead, and thinking there might possibly be some mistake I dissolved a very large quantity, without, however, discovering any trace of lead in the ore. This ore was carried by New York capitalists to Providence, where it was smelted, the zinc produced from it commanding the highest market price.

With all the mineral resources which the Virginias have, it is a matter of constant surprise to me that they are not the richest States in the Union. They can produce a good iron very cheaply, and

might, in a few years, take the coke trade away from Connellsville. With moderate energy they ought to be able in a few years, to pay the whole of their State debts in full, with back interest, without any phase of repudiation, and still be rich. I can only account for their present poverty by the shortsightedness of some of their transportation companies, and the want of energy of their people.

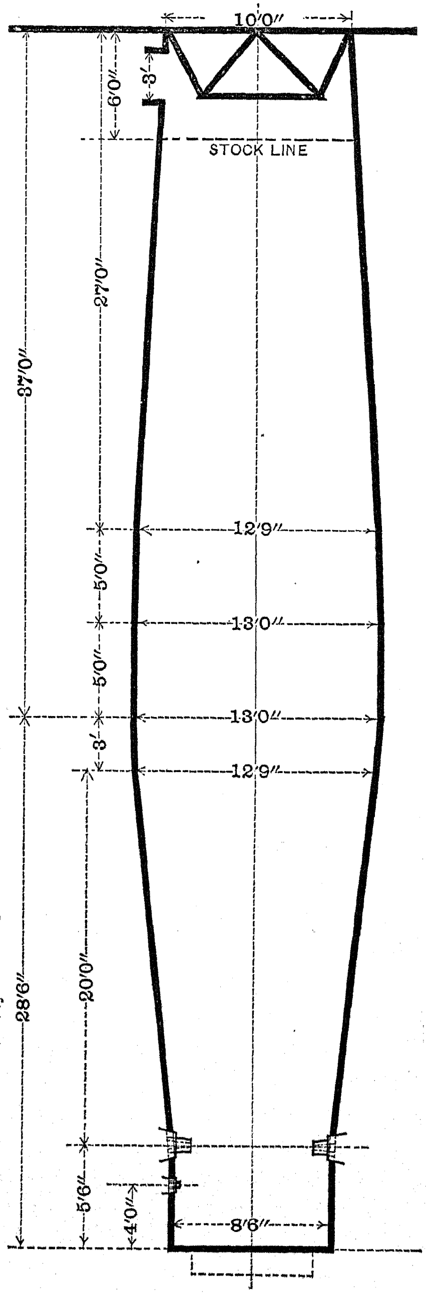
BLAST-FURNACE WORKING.

BY JULIAN KENNEDY, EDGAR THOMSON STEEL WORKS, PITTSBURGH, PA.

THINKING that it may prove of interest to the Institute, I have prepared a short account of the blowing in and subsequent working of the "A" furnace of the Edgar Thomson Steel Works. This furnace was built for the purpose of making spiegel, but has been started and is now running on Bessemer iron. The shape and size of the furnace are shown in the accompanying drawing. It will be seen that the section is formed of lines making very small angles with each other. In fact, an arc of a circle can be drawn, from the top to the tuyeres, which will nowhere deviate more than two inches from the lines given. In building, also, great pains were taken to gradually round the angles, so that the most careful observer, standing in the crucible and looking upward, could not tell at what points the slopes changed. It will also be seen that the widest part of the furnace is almost exactly midway between the bottom and the stock line. The flattest slope is maintained for 20 feet above the crucible, and is about 1⅝ inch to the foot from the perpendicular. The furnace is lined with small brick throughout, and has six tuyeres four inches in diameter, which project seven inches inside of crucible.

There are three Siemens-Cowper-Cochrane stoves, each 15 feet in diameter by 50 feet high. Blast is furnished by an upright engine with 32-inch steam cylinder, 84-inch air cylinder, and 48-inch stroke. The steam cylinder has balanced poppet valves, working horizontally. The steam valves are worked by a Porter link-movement in order to obtain a variable cut-off. The exhaust valves are worked by a separate eccentric. It is intended to attach a governor to the cut-off, but at present it is set by hand. The steam cylinder is

steam-jacketed, and will be lagged, but it is not finished yet. A Bulkley condenser is attached to the engine. Steam is furnished by cylinder boilers 42 inches diameter and 65 feet long, each having an auxiliary boiler 36 inches diameter and 50 feet long, suspended below it by legs on every other sheet. Ten of these are now up, and six more will be erected shortly, making sixteen in all, which are expected to furnish steam to run three furnaces. Water is supplied by two duplex compound Worthington pumps, with 20-inch plungers and 36-inch stroke, with Bulkley condensers attached. Two Worthington high-pressure duplex pumps, with 7-inch plungers, working in connection with an accumulator, supply water for the boilers, cranes, etc.



The stack had been pretty thoroughly dried before starting by about four months' hard firing with cordwood and coal, and the stoves also had been very well dried. I believe that a good deal of trouble in blowing in furnaces is often caused by the time-honored custom of letting them burn from twelve to thirty-six hours before putting on the blast, thus causing a large amount of cold air to be drawn in, forming clinkers on the boshes; and that difficulty is brought about also by blowing too light a blast to

keep the furnace moving at a proper rate of speed for several days afterward. Therefore I decided to burden rather light at first, and to endeavor to start off at a good round pace and try to hold it, and after a few days to come up quickly to full working burden.

The furnace was filled by putting in two lengths of cordwood on end, and two more lengths round the sides, in a layer about a foot thick, to protect the brick from the coke as it was dumped down from above. We next filled 6000 pounds of coke, putting in with it 15 per cent. of stone, no furnace cinder being used. We then put in seven charges, consisting each of 3150 pounds of coke and 1500 pounds of ore, a mixture of equal parts of McComber, Tafna, and Pilot Knob ore. We next put in six charges of 3150 pounds of coke and 2100 pounds of ore. Then followed ten charges, each containing 3150 pounds of coke, and 2400 pounds of ore. Limestone was added to each of these charges in such amounts as would give a slag of the following composition: 33 per cent. silica, 45 per cent. lime, 5 per cent. magnesia, assuming that the first iron would have 3.5 per cent. silicon in it; it had about 4 per cent., however.

This amount of stock filled the furnace nearly to the stock line; and at 9 o'clock on the evening of January 4th, 1880, it was lit with some little attempt at ceremony by the little daughter of Captain W. R. Jones. As we had no stockhouse, and it had been raining almost constantly for a week before starting, our stock was very wet, and, as a natural result, the water had worked down and dampened the wood so much that it refused to burn. A lot of live coals were shovelled in at the cinder-block opening, but in spite of all efforts it refused to draw until the next morning, when we put in more coals and turned on a light blast through one tuyere. This worked the fire through the wood in about two hours, when the blast was turned on all around, the pyrometer showing the heat of blast to be 200° F. The burden was now raised to 3000 of ore, one-sixth Somorrostro being put on in place of part of the McComber, making the proportions of ores the same as shown in the tables of materials annexed. These proportions have not been changed up to the present time. We blew for an hour with the bell open, and then closed it and lit the gas in the stoves. The blast was kept rather light till the wood was nearly all burned out and coke had come down to the tuyeres, when it was put up to twenty revolutions of the engine. The temperature of the blast rose steadily at the rate of about 70° an hour. We got the first cinder at the bottom of the furnace on the morning of Tuesday, January 6th; and at 8 o'clock on Wednesday morning, January

7th, iron appeared at the cinder tap. It was then tapped out below, giving a cast of twenty-two tons of No. 1 metal. On Thursday, the 8th, the ore burden was raised from 3000 to 4240 pounds. From this it was gradually raised till, on Thursday, January 15th, it reached 5460 pounds. The subsequent rate of increase was very gradual, until it reached 5760, as shown in the table.

For the first three weeks the slag was kept at about the composition—33 per cent. silica, 45 per cent. lime, and 5 per cent. magnesia, giving iron with about 2.4 silicon and never more than .003 sulphur. But owing to the fact that the converting works had a large amount of iron on hand which was very high in silicon, we changed our mixture so as to produce a slag of the composition given in the table. From this we get iron with about 1.7 per cent. silicon and .02 per cent. sulphur.

The first week after starting we made 442 gross tons, 1990 pounds ($442\frac{1990}{2240}$ tons); the next week, $506\frac{1060}{2240}$ gross tons. During this week the furnace worked a good deal on one side, and always hung after casting till the blast was taken off to let it settle. On Sunday, January 25th, the furnace hung badly and went off on black cinder. 15,750 pounds of coke were charged blank, under the influence of which normal working set in on Monday morning, and $528\frac{756}{2240}$ tons were made that week, notwithstanding a stoppage of four hours to make connections from main steam pipe to Furnace "B" boilers, and five hours to connect the blast receiver to Nos. 2 and 3 engines. From this time the furnace ran along steadily till 6 o'clock on the morning of Saturday, February 14th, when the high water broke into our stock-house and prevented the fillers from working. At 8 o'clock we cast, stopped the tuyeres, and suspended operations till 4 o'clock P.M. on Sunday, February 15th, when the water was low enough to start. We found that the temperature of the blast, which had been 1000° F. when we stopped, was 850°. No extra coke was charged, and the engine was started at the same speed at which it had been running before we stopped. The temperature of the blast came up to 1050° in two hours, and the furnace ran off just as if there had been no stop, at an 85-ton gait. In the following table is shown the composition of the materials used, and also of the products:

MATERIALS.		COMPOSITION, PER CENT.							
Name.	Weight per charge.	SiO_2.	Al_2O_3.	Fe.	Mn.	CaO.	MgO.	P.	S.
Tafna ore	1920	3.28	3.86	58.65	0.97	2.05	0.01	0.044	0.068
Pilot Knob	1920	14.10	3.05	58.69	......	0.16	......	0.039	0.077
McComber	960	11.68	0.65	52.41	3.52	0.62	trace.	0.036	......
Somorrostro	960	8.12	1.02	49.15	1.17	6.39	0.56	0.012	0.030
Limestone	2100	11.36	2.35	......	......	42.97	4.66	0.009	......
Coke	3150	6.00	3.00	......	......	......	......	0.010	1.250
PRODUCTS.									
Slag	......	38.71	11.61	......	......	41.30	5.04	......	1.670
Metal	......	Si. 1.72	......	......	......	......	......	0.072	0.021

The yield of the furnace has been as follows in gross tons:

Wednesday, January 7th,	$35\frac{1060}{2240}$
Thursday, January 8th,	$32\frac{860}{2240}$
Friday, January 9th,	$40\frac{1370}{2240}$
Saturday, January 10th,	$48\frac{360}{2240}$

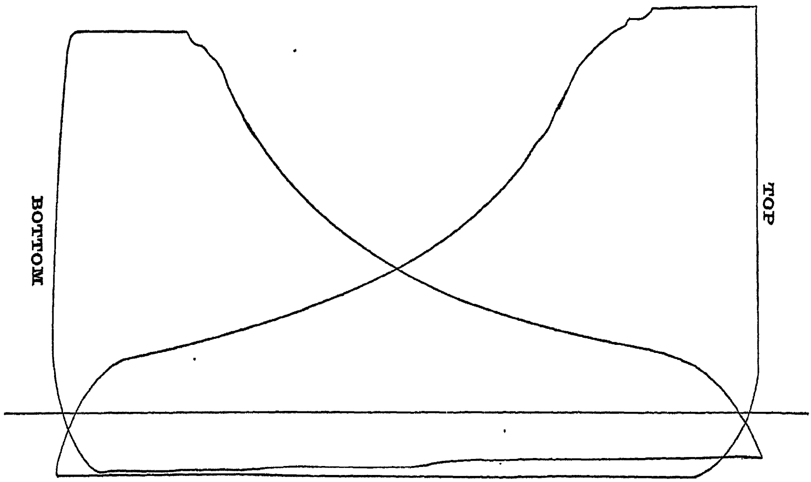
Beginning with the first full week, the coke used and metal made were as follows:

DATE.	Coke, pounds.	Metal, gross. tons.	Coke to 1 lb. of iron, lbs.
Week ending Saturday, January 17th	1,168,650	$442\frac{1990}{2240}$	1.180
Week ending Saturday, January 24th	1,156,050	$506\frac{1060}{2240}$	1.019
Week ending Saturday, January 31st	1,253,700	$528\frac{756}{2240}$	1.059
Week ending Saturday, February 7th	1,222,200	$537\frac{450}{2240}$	1.015
Six days ending Friday, February 13th	1,043,050	$480\frac{330}{2240}$	0.970

Ore used for four weeks and six days ending February 13th .... 10,116,860 pounds.
Actual yield of ore for that time .... 55.2 per cent.

Of the production up to this time, according to the steel-works grading, over 98 per cent. was No. 1, and the balance Nos. 2 and 3. Except on the Sunday when the furnace went on black cinder, no firing has been done at the boilers. As there is no roof over the boilers and the tops of them are not covered yet, and as there is about 300 feet of 20 and 15-inch steam pipe which is not lagged at present, the consumption of gas under them is pretty large. When everything is completed I have no doubt that we will be able to carry 1200 degrees temperature of blast, with a corresponding decrease in amount of coke required. The average temperature of the blast has been 1050° F.; the average pressure, $7\frac{3}{4}$ pounds at engine and $6\frac{1}{4}$ pounds at tuyeres. The somewhat excessive loss of pressure is

partly explained by the fact that the pipes, stoves, etc., were proportioned on the basis of a production of 50 tons a day. The temperature of the waste gases remains with remarkable uniformity at 340° to 350° F., except when the materials are very wet, when it runs down



70 lbs. steam-pressure; 6¼ lbs. blast-pressure; 11 lbs. vacuum;
32 in. steam cylinder; 84 in. air cylinder.

as low as 250° F. Not less remarkable is the constancy of composition shown by the gases, which for the last ten days have run from 12.2 to 12.5 per cent. by volume of CO_2, and from 27.2 to 27.8 per cent. by volume of CO,—average ratio, 0.447.

I show a pair of diagrams taken from one of the blast engines, not as an example of perfect diagrams by any means, but to show that a direct-acting slow-speed engine can be run with a comparatively short cut-off. We have tested our engines by running them at 14 revolutions and cutting off at one-tenth stroke, and find no signs of dragging on the centres. While these engines are not as economical as a good compound engine, they are more economical than the average blast-furnace engine used in the region around Pittsburgh, and are a step in the right direction. It seems likely that the future improvement in blast-furnace construction will be largely in the direction of more economical blowing machinery; and, notwithstanding the conservatism of blast-furnacemen, it will not probably be very many years before blowing engines will be designed to do as high duty as the best pumping engines, and the fact will be appreciated that *heat* is money and should not be wasted un-

necessarily even around a blast furnace. The working of this furnace suggests some very interesting questions as to the relative economy of large and small furnaces; but as we expect to have a 20 x 80-foot stack in operation shortly right alongside of the 13 x 65-foot stack, I will not pursue the subject further at present, but may, at some future time, be able to give some information on it from observations of actual working.

NOTE.—The record of the furnace for March, which I have the opportunity of adding to this communication, is as follows: Total yield for the month 2762½ tons. As one day was lost from causes outside of the furnace, this gives a weekly average for thirty days of 644½ tons. The best week's work was 671 tons, and the best day's work 114 tons. The yield of the ore for the month was 54.4 per cent. of iron, and the amount of coke used was 1.029 pound to the pound of iron.

DISCUSSION.

MR. BIRKINBINE asked Mr. Hartman how the lines of the Edgar Thompson furnace corresponded with the formula proposed by Mr. Bennett.

MR. HARTMAN: The dimensions proposed by Mr. Bennett depend upon the diameter of the hearth as a basis. The comparison, with this dimension as a starting-point, is as follows:

	Bennett's.		Edgar Thomson.	
	Feet.	Inches.	Feet.	Inches.
Hearth,	7	9	7	9
Height of bosh,	21	9	25	6
Bell,	5	6	7	0
Tuyere circle,	6	1	7	4
Height,	58	0	65	0
Bottom to centre of tuyeres,	4	0	5	6
Area of annular space between bell and wall,	24	0	49	0

On the drawing the diameter of the hearth is given as 8 feet, 6 inches. This is owing to the fact that the walls were carried down vertically. If the bosh line had been continued as was originally intended, the diameter would be found to be 7 feet, 9 inches, at the bottom, as above given.

The lines of the Edgar Thomson furnace were fixed by Mr. William P. Shinn (then general manager of the company) and Taws and Hartman without any reference in any way to Bennett's lines.

David Thomas and his sons on the Lehigh tried large hearths in 1849, and finally, at Hokendauqua, in 1858, established 10-foot hearths for 18-foot bosh, while Truran, at Dowlais, England, was the first to successfully introduce wide tops in 1851.

THE PUDDLING PROCESS, PAST AND PRESENT.

BY PERCIVAL ROBERTS, JR., PHILADELPHIA.

It may seem necessary to offer an apology for presenting for consideration a process which is conspicuous by its absence in the literature of the Institute, and which may be thought by some to belong to the past in metallurgy, and to have been already superseded. But the large capital invested in puddling calls for a careful consideration of the question whether the time has certainly arrived when the puddling furnace must be replaced by the converter and open-hearth furnace. May there not still be a place for puddled iron alongside of molten iron and steel, and is not the improvement of the puddling process itself worthy the attention of engineers equally with the Bessemer and open-hearth processes?

The changes involved in the conversion of pig iron into wrought iron are well understood and need only be briefly alluded to. The patent of Henry Cort bears the date of 1784. Since that time the improvements in the process have mainly consisted in the replacement of sand by iron bottoms by Samuel Baldwyn Rogers in 1818, and the still more recent substitution of iron oxide for the refractory materials used for the sides and bridge of the hearth, which distinguishes the wet or boiling process from the dry or puddling process. Chemically, the process consists in the removal of the metalloids from the pig iron, a result effected mainly by the iron oxide. Silicon is first oxidized, then the phosphorus, and finally the carbon. The silicic and phosphoric acids produced pass into the cinder and the carbonic oxide burns as it escapes from the bath of metal.

It is interesting in this connection to note the effect of temperature on the removal of the phosphorus from the iron. As is well known, no phosphorus is eliminated under the oxidizing influences prevailing in the Bessemer converter, while from 70 to 80 per cent. is removed in puddling. But we find, if in working cold short irons

the temperature of the furnace is much increased towards the end of the process, that a considerable amount of the phosphoric acid is deoxidized and phosphorus again combines with the iron. This reverse process is aided by a siliceous cinder arising either from the use of a very siliceous pig iron, or of an over-siliceous ore for fix. The fact which has been known for some time that only a basic cinder can retain phosphoric acid has given rise to the "basic lining" which now attracts so much attention in the Bessemer process. For the conditions affecting the removal of phosphorus from pig iron I would refer to the careful and complete experiments of I. Lowthian Bell, in England.

Notwithstanding the recent progress in the metallurgy of iron the puddling process is essentially what it was three-quarters of a century ago,—laborious, crude, and unsatisfactory. The attempts at improvement in the process may be classified under two heads: 1. economy of labor; and 2, economy of fuel.

Increase of yield and improvement of quality are so intimately connected with both of these two classes that it is not easy to consider one apart from the other.

1. Economy of Labor.

For the successful accomplishment of the operation of puddling it is necessary to bring the molten metal into contact with the solid oxides by agitation effected either by human or mechanical agency upon a stationary hearth, or by giving motion to the whole body of the furnace. One of the first attempts for lessening the labor of the puddler is recorded in a drawing at Dowlais which has been traced back to the year 1834. It is a reverberatory furnace with a revolving hearth, driven with a vertical shaft by bevel gearing. Whether this machine was ever used I do not know, but it is of interest as showing that most of subsequent improvements are not new in principle. Coming to more recent times, we have the Richardson process of blowing air into the molten bath through a tubular rabble. The advantages claimed for this method are that it hastens the boil, reduces the labors, and produces a tough metal of uniform and high quality. After the iron has come to a boil the rabble is withdrawn and the working continued in the ordinary manner. I believe this process has never been used in this country and but sparingly in England.

Morgan's puddling machine consists of a reverberatory furnace of

the usual form, which has an opening in the roof through which a vertical shaft is lowered with a horizontal arm. The shaft is set in motion by suitable machinery and the arm revolves in the furnace, doing away with the labor of the puddler and helper until the heat is ready for balling, when the shaft and arm are withdrawn, the opening in the roof closed, and the balling proceeded with in the usual manner. The wear and tear connected with this method must be enormous, and the results, I should think, not very satisfactory.

Griffith's and Whitham's devices are similar in idea but different in mechanical details. Their object is to give an oscillating movement to a rabble of the ordinary shape by means of machinery, the puddler or helper merely guiding the rabble. The balling is accomplished in all cases by hand labor. None of the above-mentioned improvements do away with the skilled workman but merely lessen the laborious work of the early stages of the heat, which requires brute force rather than experience.

In a work by Kohn upon the *Manufacture of Iron and Steel* will be found more detailed statements concerning these processes. That any of them has proved satisfactory, I question. One of the imperfections common to them all is the difficulty of keeping the raw iron from gathering in the crevices of the fix and settling on the bottom and in the corners of the furnace into which the rabble does not enter, leaving the furnace at the conclusion of the heat in a very dirty condition. We all know the importance of a thorough working of iron in the jambs of a furnace, as it is there that the metal begins to gather when coming to nature, requiring careful working for good results. Another serious objection which may be advanced against these processes is, that they require the same skilled workmen to operate them as are needed for the old style of hand puddling. No increase in the number of heats is obtained, for the men, instead of encouraging experiments, look upon them with great distrust as inimical to their best interests, and when a workmen and his tools do not agree good results cannot be expected.

About 1867 a change in the direction of improvement took place, and it was reserved for an American, Samuel Danks, to have the boldness to propose an entire revolution in the puddling process. The Danks furnace was the first rotary furnace to be put into successful operation, although its success was not assured until many improvements and alterations were made upon the original designs.

In England this same idea was elaborated, and several machines were brought out differing in details. The one of most novel con-

struction was the Godfrey-Howson furnace, which has but one opening into which the heat enters and the products of combustion escape, a blowpipe on a large scale being substituted for the ordinary fireplace. Later, in this country, we find a rotary furnace designed by the Edgemoor Iron Company of Wilmington, Delaware, worthy of mention from the fact that this company is at present equipping their works with these furnaces, which would seem to indicate great confidence upon the part of the proprietors in the success of the rotary process.

2. Economy of Fuel.

In the utilization of coal for puddling two methods are employed. The one in almost universal use, where coal is directly burned on the grate of the furnace, is irrational and wasteful. The other method, consisting in the conversion of the coal into combustible gases, which are burned on the hearth of the furnace, though more economical and rational, is but seldom used.

The attempts which have been made to improve the old system may be divided into two classes: First, those having for their object the prevention of smoke by feeding the coal below the surface of the fire, which is always kept bright. The mechanical devices for accomplishing this object are found in the Frisbie & Sweet furnaces. The system has not come into general use. An objection in the case of coal forming clinkers is, that the clinkers are forced to the top of the fire. Second, those having for their object the utilization of the volatile matters of the coal by a partial coking of the coal before it reaches the fire. This is effected by the employment of a separate magazine in connection with the fireplace. The gases from the coal are caused to pass over the fire and are there burned. Of this variety of furnace may be mentioned the Wilson furnace, and of more recent date the Price furnace, which has given very good results.

When we consider, however, the cost of introducing these improved furnaces, and the trouble and annoyance of teaching workmen to use them, it is evident that we might just as well go a step further and introduce the gas system in its entirety. The great advantages to be gained in the use of gas in puddling are well known. We may distinguish here two systems, the continuous-acting furnace, of which Swindell's furnace is an example, and the well-known Siemens regenerative furnace.

Of the use of water-gas in the place of the ordinary generator-gas it is too soon to speak, but reference may be made in passing to the

astonishing results said to have been obtained at Washington by the Gill process, with gas containing as high as 75 per cent. of combustible gases, which we take *cum grano salis.*

What, let us now ask, is the present state of the puddling process, and what relation does its welded product sustain to the fused product of the Bessemer converter and the Siemens furnace? Will steel supplant iron?

In a paper on the Separation of Phosphorus from Pig Iron, read before the Iron and Steel Institute of Great Britain in 1878, by I. Lowthian Bell, occurs the following:

"The elimination of this metalloid from pig iron is, doubtless, a subject of great interest and importance to British smelters, having regard to the fact that nearly five-sixths of the metal obtained from their native ores contains so much of this impurity as to unfit it for the manufacture of steel, that form of iron which bids fair to supersede, in a great measure, the product of the puddling furnace."

If phosphorus cannot be removed, the question is easily settled; the production of steel is a limited one; and in the future, as at present, it will be made from the highest grades of our pig irons, and be used for certain special purposes, such as rails, etc., for which it has shown its great superiority over iron.

But, no doubt, many will at once say: phosphorus can be removed; the Thomas & Gilchrist process, with its basic lining, has overcome this difficulty. That phosphorus has been removed experimentally, there can be no question; that it has been expelled successfully, from a commercial point of view, is open to doubt. Of the three processes established for its elimination, the Bell, the Krupp, and the Thomas & Gilchrist processes, the second has, from an economical standpoint, produced the best results.

There are some points in regard to all of them which, in the published results of experiments, have not been very fully touched upon, though they are of great importance. Is the increased cost of working greater than the difference in price between inferior brands of pig iron and those suitable for steel-making? This is, of course, a secondary consideration if the demand for such pig exceeds the supply, but it will be of vital importance if the reverse is the case. Is the removal of phosphorus uniform or does it vary, giving us results differing from day to day? What is the percentage of bad blooms made by these processes as compared with the usual method of working? How uniform is the quality of the final product as furnished to the consumer? It will, perhaps, be said that suffi-

cient experience has not yet been obtained to answer these questions, but until they are disposed of we must be very cautious in accepting the announcement made by inventors or operators to the effect that success has been achieved. Granted that a method of dephosphorization may be established upon a commercially successful basis (and present indications seem to point toward such a conclusion), what will be the resulting product, and how well will it be fitted for its intended uses?

In advocating the use of high qualities of steel, and enumerating the advantages to be gained by employing it, the fact is frequently lost sight of that this superior metal is made from the highest grades of pig, obtained with the greatest care from the purest ores, and that the succeeding processes are worked out with the aid of the most improved plant. The metal is followed through all details of manipulation with the most thorough inspection and rigid chemical and mechanical tests. Material thus obtained is compared with wrought iron made from anything and everything. No chemist mixes the charge or analyzes the product, but a puddler is left to guard the interests at the most vital stage of the process. It is his aim to produce the greatest weight, with the least labor, in as short a time as possible, and with such work no one can blame him. It is not astonishing that under such conditions iron is so much inferior in its physical qualities to steel. Even taking the same grade of pig metal for the manufacture of wrought iron as is now used for steel, the mild grades of the latter suitable for structural purposes will, no doubt, give higher results by mechanical tests, but the difference between the two will not be as great as many are apt to think.

On the other hand, if in the future, by means of dephosphorizing processes, we shall use all sorts of pig iron for steel, shall we not introduce a dangerous element of uncertainty into its manufacture which we do not have to deal with at present? When it is considered how very slight a change in the percentage of some foreign substance may produce a considerable variation in the quality of steel, uniformity in a metal derived from such impure raw materials must be difficult to attain. The homogeneous nature of steel, as compared with the many-pieced structure of iron, is claimed as one of its advantages. Homogeneity in steel may be a cause of weakness, and the lack of homogeneity in iron a source of strength. A steel bloom, to all external appearance perfect, may be within entirely bad, either from piping in the moulds, or from other causes of a similar nature. Chemical analysis will not show this defect, and a bar

produced from the same although sound as far as can be seen, may fail in service suddenly and without warning. On the other hand, the possibility of a wholly bad iron bar diminishes just in proportion as the number of pieces in the pile from which it is made increases.

For a material for structural purposes, the term uniformity should take the place of homogeneity. A material exposed to abrasion, such as a rail receives, requires the latter quality, but one subjected to strains of compression and extension, torsion and bending, wants uniformity more than any other property. If one bad member is contained in a structure, the strength and homogeneity of the whole is of no avail. For many purposes in construction, steel may be used to very good advantage, notably for members liable to wear and parts running in bearings. But whenever it is applied in parts of varying outline, where sudden changes in form take place, planes of weakness are developed at all those points at which anything like a corner occurs, unless large fillets are used and great care is taken. It must not be forgotten that the structures hitherto erected of steel have been, as it were, experimental, and have therefore been put up with the closest inspection and caution. If it should be generally adopted, this same care could not be exercised unless an entire revolution in existing modes of manufacture takes place. The rough handling which iron for structural use receives in manipulation would be fatal to steel. Existing plant and methods of working must be abolished, and workmen be educated in the proper handling of the new products.

Looking upon the above objections as a few of the more important ones yet to be met before a more general use of steel can take place, it will be apparent that its substitution for wrought iron will be very slow and gradual. The puddler and his furnace yet have many years before them. No one could regret it more than the writer. No other process in iron metallurgy requires so much work per ton of metal produced. It seems absurd to think that the labor of two men for ten hours is necessary to produce a ton of wrought iron, and that for one ton of pig iron used one ton of coal is consumed! It is not worth while to consider those methods which aim merely to lighten but do not do away with the labor of the puddler. They may have some advantages, but they will never come into general use. What is needed is a method which is governed by intelligence, but which requires only ordinary labor for its working. The rotary furnace process is the only one which at present aims at this result, but its complete success is open to doubt. The wear and tear of the

complicated mechanism and revolving surfaces is a source of expense, and the lining is composed of a material not well calculated to resist heat. The quality of the iron, however, is good, and counterbalances many of the attending disadvantages, although it will not, as was at one time hoped, answer for making bars without weld. It must be cut and piled as ordinary iron, or the work upon it will not suffice for good results.

We are now in the midst of an epoch of uncertainty; a few years more and the success or failure of steel to supplant wrought iron will be established beyond a doubt. Its success depends upon the results which shall be obtained from the working of all grades of pig iron; and its failure is certain if uniform quality cannot be produced. For the present, therefore, the system of puddling must continue as of old; but every ironmaster, not only of this but of other countries, will most gladly welcome the process, whether it be of steel or of iron, which shall do away with the weary toil of so many thousands, and usher in a brighter and a better era than could ever be accomplished by the puddling process as invented by Henry Cort.

NOTES ON BATTERY AND COPPER-PLATE AMALGAMATION.

FROM THE MINING LABORATORY OF THE MASSACHUSETTS INSTITUTE OF TECHNOLOGY, BOSTON.

BY ROBERT H. RICHARDS, S.B., PROFESSOR OF MINING.

VERY little has been published recently on this subject in the mining journals or proceedings of societies. The attention of experts has been diverted perhaps by the demands for pan amalgamation of refractory ores. Yet free milling ores occur in many localities, and it may be interesting to have on record some data as to details of running a small test-mill, especially as details of results do not seem to be accessible to students and investigators.

Küstel, in 1863, under battery amalgamation, dismisses the subject by stating that "amalgamation on copper-plated platforms, troughs, and other copper fixings" is " very imperfect, and mostly abandoned." J. Arthur Phillips says, in 1867, under placer mining: "A well-amalgamated copper plate is considered as effective for

saving fine gold as an equal surface of pure mercury." In Dr. R. W. Raymond's reports, I find in two volumes mention of amalgamation by copper plates.

In 1871 N. S. Keith, in an admirable article on the preparation of amalgamated copper plates, says: "The discovery of the utility of amalgamated copper plates in the treatment of auriferous rock in the stamp-mill has so simplified and cheapened the metallurgy of gold that it is now profitable to mine and reduce many gold-bearing ores and rocks that formerly would not pay." In Dr. Raymond's report in 1872, J. A. Church discusses the use of copper plates with reference to his theory of amalgamation. I am unable to find any allusion to amalgamation in either Rittinger's or Gaetzschmann's excellent works on *Aufbereitungskunde.*

Our experience at the Massachusetts Institute of Technology has been as follows:

In the winter of 1872 the little five-stamp battery was started in good earnest. Small lots of ore, severally from North Carolina, Georgia, Colorado, and New Hampshire, were run in the mill and over the plates. After each run, the plates were cleaned up, and the amalgam retorted. Our tailings were not as clean as we could have hoped, and the yield of gold, therefore, was not so high as we expected, for causes which will be partially accounted for later on in this paper.

After the last of these runs, the plates were scraped a second time, and, to our dismay, some more gold was obtained; a third time, and more gold. We began to wake up to the fact that we had been crediting North Carolina ore with Georgia gold and so on. This difficulty became so apparent, and seemed so insurmountable, that we wellnigh despaired of ever using the mill in making careful quantitative tests. I shall speak of this difficulty, in the course of this paper, as the *overlapping error.*

Until 1876 we did nothing more with the mill, owing to other cares. During that year, however, we tried a new expedient for overcoming the overlapping error. It was to purchase the thinnest copper that could be found in the market, about $\frac{1}{50}$ inch, and to use a new set of plates with each run, dissolving the old ones in sulphuric acid at the end of the run, and thus recovering the gold. We got back the gold, to be sure, but after much trouble; and in this way we entirely overcame the overlapping error. We recovered, after running 1500 pounds of $14 rock, besides the amalgam which we

scraped off, about $2.75 of gold which had soaked into the plate, 10 feet long, 6 inches wide, or 5 square feet of surface. Still the tailings were too rich, and the mill work was not altogether satisfactory. We finally found that, in avoiding the overlapping error by the above expedient, we had run into another and, possibly, a worse error, which I shall call the *new plate error.*

Mill men will tell one that no mine is willing to send ore to a mill that has just put in new plates; that the new copper plate, coated with mercury, is not well calculated to catch gold; that a layer of gold amalgam must be formed on the surface before the plates can be considered at all reliable. In fact, we are informed that, in the Black Hills and elsewhere, a mill with new plates counts it as one of its regular expenses to buy 100 tons or so of gold ore and run it, to charge the surfaces of the plates with gold amalgam, before taking in custom work. Recently we have adopted the suggestions in N. S. Keith's paper (Raymond's Report, 1871). These, with certain modifications, have overcome all the difficulties, and given us the means to test gold ore by the ton accurately and efficiently.

Our method is to take soft copper plate, $\frac{1}{16}$ inch thick, of the size required. Scrub it with sand, sal-ammoniac, quicklime, and mercury, until it is plated. Dilute nitrate of mercury will plate the copper far quicker and with less trouble, but not so permanently, so deeply, or so thoroughly. If very obstinate spots are found, dilute nitrate of mercury may be used; and if this fails, a plumber's scraper will remove the spot. When thoroughly plated and washed clean, the plates are dried carefully with cotton batting. Mercury is then sprinkled on through chamois-skin, and distributed with cotton. This will give the plates a brilliant, mirrorlike surface, which will remain for several days if water is kept off them. The least addition of water will very soon bring out the yellow stain that is the enemy of the amalgamator. This yellow stain is prevented on old plates by the coating of gold amalgam which has formed on their surfaces.

We have hit upon an expedient for overcoming the new plate error and the formation of the yellow stain which so much retards the amalgamation of gold. Our device is in the substitution of silver amalgam for gold amalgam. Silver amalgam is made by dissolving silver dollars in nitric acid diluted with an equal weight of water, and then adding about ten times as much quicksilver as silver. The silver is all precipitated, making a pasty amalgam; the liquor, after standing over night, is then decanted, and the remaining amalgam washed thoroughly with water. One silver dollar will make amal-

gam enough to paint about 4 square feet of surface. This amalgam is dried and squeezed in chamois-skin, to remove the excess of mercury, until about the consistency of soft putty. It should then be painted on dry to the bright plates with a flat hog's-bristle paste-brush. The amalgam should be allowed to drain for at least twelve hours, to avoid, as far as possible, the presence of free mercury on the plates during the run, as it is liable to flour and be lost.

Silver amalgam is just as good a catcher of gold as the gold amalgam, so far as we yet know. This layer of silver amalgam also overcomes the overlapping error, because the gold never comes to any considerable extent into contact with the copper plate; and hence, when the plates are scraped off in the usual way, the silver amalgam practically carries off with it the whole of the gold. For a testing-mill, the method is sufficiently easy and simple, and has been adopted by us as overcoming all the difficulties yet encountered.

The Institute mill is a single five-stamp battery, with stamps weighing 55 pounds originally (now worn down perhaps to 53 pounds), $3\frac{1}{2}$-inch face of shoe and die, $9\frac{1}{2}$-inch drop, 88 drops per minute. It uses about 34 kilos of water per minute, and stamps through a screen punched with holes $\frac{1}{50}$ inch in diameter. The discharge is on both sides—that is, double issue. We amalgamate in the battery without battery plates. The plates are five in number; net length available for amalgamation, 5 feet 6 inches each, or total length $27\frac{1}{2}$ feet by 10 inches wide. The plates slope $1\frac{3}{8}$ inches to the foot, and have never given much trouble from settling sand. This is the minimum slope at which we should dare to place them; $1\frac{1}{2}$ inches to the foot would be a better slope. The mercury trap at the end of the plates is a kind of quicksand apparatus, which allows the mercury to discharge below as fast as it accumulates.

When cleaning up, the battery amalgam is kept by itself, and each one of the five plates by itself. This enables us to see when and where the amalgamation takes place. The sand is separated by a spitzkasten apparatus into sand and slimes. The sand is concentrated, and yields concentration and tailings. Assays are then made of the original ore, the residue in the battery, concentrations, tailings, and slimes. When an ore has been subjected to this series of tests, we know, to say the least, a good deal about it.

TABULAR STATEMENT OF THE PROCESS.

Ore, Mercury, Silver,
- Amalgams,
  - Battery, 1st plate, 2d plate, 3d plate, 4th plate, 5th plate, Mercury trap. — Each parted by itself.
- Sand or spitzkasten,
  - Slimes, for chlorination or other treatment.
  - Sand on table,
    - Concentration for chlorination or other treatment.
    - Tailings—waste.

Pure mercury is used in the battery, and hence this lot when assayed gives us the fineness of the gold in the ore.

Besides the above tests, a sample is always taken from the pulp as it comes from the stamp-mill, and also as it comes off the plates. These two samples are taken at the rate of ¼ liter dipperful every fifteen minutes. They serve to give additional testimony in regard to the working of the plates.

The rate of stamping rock has not been satisfactorily settled, as the mill was out of line about half an inch at the time of the runs this year. The shoes overlapped the edges of the dies half an inch. When New Hampshire quartz was fed, in size about 1-inch cube, the best rate maintained was 54 kilos per hour. When this rock was crushed by rolls through a ¼-inch sieve, it then stamped 69.6 kilos per hour. These figures are, however, only approximate averages, as they depend largely on the skill of the feeder. Either overfeeding or underfeeding lessens the yield.

Four tests have recently been made; two of them on a low-grade ore from New Hampshire; the others an ore from Nova Scotia, which was sent to us through the kindness of Mr. J. Fraser Torrance.

I. The New Hampshire lot. This was composed mostly of quartz, stained yellow with iron rust, having cavities from which the iron pyrites had been dissolved. It contained a small percentage of pyrites, but no gold that was visible as we examined the heap. This lot also contained some fine mica-schist, which was probably from the wall-rock.

This lot of ore weighed 2021.6 kilos (4447.5 pounds). A sample of it yielded to fire assay $4.14 in gold to the ton of 2000 pounds, reckoning gold at $20.67 per ounce troy. The total value, therefore, of the gold in this lot was $9.20.

The yield of gold in the different parts of the process was as follows:

Battery amalgam,	$4.573
1st plate,	2.694
2d plate,	.215
3d plate,	.180
4th plate,	.099
5th plate,	.050
Mercury trap,	.038
Total yield,	$7.849

Samples were assayed for gold as follows:

	Per ton.
Pulp sample as sand went to plates,	2.07
Pulp sample as came off the plates,	trace.
Pyrites concentrations weighing 74 kilos,	$5.17
Coarse tailings,	trace.
Fine tailings or slimes,	trace.
Residue in battery, weighing 11.75 kilos,	$17.57

The result of this experiment may be summed up as follows:

	Per ton.
Gold by fire assay,	$4.14
Gold extracted by amalgamation,	3.53

or 85.2 per cent. is saved by amalgamation.

The poverty of the iron pyrites concentrations found by us corroborates the conclusions arrived at by Professor Hitchcock, who says that he has universally found the gold in these mines to be in the quartz and not intimately associated with the pyrites, as it is in some other localities.

II. The second lot of ore was taken from the same heap as the last, but was taken at a different time. The lot weighed 403.06 kilos (888 pounds), and assayed $4.14 per ton. When cleaned up, the different parts yielded gold as follows:

Battery amalgam,	$1.220
1st plate,	.430
2d plate,	.063
3d plate,	.041
4th plate,	.028
5th plate,	.020
Mercury trap,	.020
Total yield,	$1.822

or $4.10 per ton.

According to the above assays the lot yielded to fire assay $4.14, to amalgamation $4.10 per ton, or 99 per cent. of the gold was saved. This, of course, we cannot claim to have done; we can only say that the method of valuing by fire assay is not fine enough to give the cents exactly on such low-grade ore.

III. The third lot of ore was from Nova Scotia, and was composed of bluish-white quartz and blue slate. It contained a very little pyrites, arsenical pyrites, and galena. The gold sights, or visible lumps of gold, had all been picked out of this lot of ore; and when it was crushed in a Blake's crusher to 1 cubic inch in size, it showed no gold. These sights, I should judge, may have amounted to as much as $5 or $6 for the lots III and IV inclusive.

This lot III weighed 639.87 kilos (1407.7 pounds). No satisfactory valuation could be obtained from it, as the gold proved to be in such coarse condition. A sample of 48.45 kilos (106.6 pounds) when crushed and passed through a sieve having 12 holes to the linear inch yielded:

	Per ton.
Gold pellets on sieve,	$7.10
Siftings, by fire assay,	7.23
Total value,	$14.33

After the ore was stamped and the mill cleaned up the following figures were obtained from the different products:

Battery amalgam,	$9.1324	gold.
1st and 2d plates together,	.1389	"
3d plate,	.0083	"
4th plate,	.0188	"
5th plate,	.0048	"
Mercury trap,	.0134	"
Total,	$9.3166	gold.

or $13.24 to the ton.

The amalgam from the battery was in large, coarse nuggets, attached together by liquid mercury. Scarcely any of this gold appeared to have made its way through the sieves. A very different result is shown in both the first and second lots of ore, where the gold was evidently fine, and therefore passed through quite freely, and more than one-third was caught on the plates.

It should be stated here that the lot marked No. II was run after that marked No. III. By comparing the yields of the plates of No. I, No. III, and No. II, it will be readily seen how perfectly

we have overcome the overlapping error. Samples were assayed as follows:

	Per ton.
The pyrites concentration, weighing 9.9 kilos,	$10.33
Coarse tailings,	trace.
Fine tailings,	trace.
Battery residue, weighing 7.05 kilos,	$6.20
The sample of pulp fed to plates,	trace.
The sample of pulp from off the plates,	trace.

IV. A second portion of the above lot was stamped without the use of amalgam or amalgamated plates. This lot weighed 631.8 kilos (1390 pounds). When the battery residue was panned out, it yielded nuggets of gold valued at $8.24, or $11.85 per ton. The yields, therefore, by the above tests were: by fire assay, of doubtful accuracy, $14.24; by stamping with amalgamation, $13.24; by stamping alone through $\frac{1}{50}$-inch holes, $11.85. The tests of each of these two lots of ores were carried on under the supervision of a mining student, who was assisted at his work by the rest of his class.

The following precautions in managing gold-mills have been mostly obtained by conversing with mill-men from different parts of the country. Our experience entirely confirms the truth and importance of them:

The amalgam on the plates should not be allowed to get too hard or too soft. If too hard, it may fail to catch the gold; if too soft, mercury will flour at the lower end of the plates.

A quantity of mercury, varying from 1 to 1½ ounces, should be fed to the stamps for every ounce of gold contained in the ore under treatment. This mercury should be fed a little at a time every half hour. If the plates are inclined to stain, a lump of cyanide of potassium held a moment on the spot will remove the yellow stain. Too free use of the cyanide will cause the amalgam to soften too much on the plates, and cause a loss at the lower end.

Keith says that if an ore contains ferrous or cupric sulphate (blue or green vitriol), as a result of oxidation of pyrites, giving rise to the yellow stain on the plates, a little lime should be added along with the ore. This will precipitate the metals, forming harmless gypsum with the liberated sulphuric acid.

The use of oil about the mill should be restricted as much as possible, as this, more than anything else, causes the yellow stain to come. If the cams are oiled with an oily rag on a stick every half hour, the excess and dripping may be largely prevented. Molasses is sometimes used to lubricate the cams, but it is a poor lubricant.

It does no harm to the amalgam. The cams should be wiped clean every day.

It would seem to us that the method adopted in our laboratory is equally well suited for large mills. The first two cleanings-up would be diluted with silver as a matter of course; but after this, no such trouble would ensue.

Copper plates in the battery we are inclined to recommend strongly, although we have not used them ourselves, owing to the difficulty of making a quantitative experiment with them.

Thus far, our search for methods for testing such small quantities of gold in tailings and products as we obtained in these runs has been fruitless. The chlorination method gives the greatest encouragement; but this is a long and tedious assay, and has to be performed with the utmost care.

The fire assay is of little use, as the quantities taken are too small to give any result at all on $1 tailings. We are just now working up an amalgamation method for tests of tailings, which gives some degree of promise, and which, if successful, we hope to describe at some future date.

The development of this method of testing gold ores is largely due to the energy and zeal of the several assistants in the mining laboratory during the last eight years, namely, Messrs. William Foster, E. Faunce, B. P. Tilden, and R. W. Lodge, who, together with the students of the successive classes, have spared no pains or trouble to make the laboratory methods a success.

DISCUSSION.

DR. EGLESTON: My experience is that all the gold mills in the West which can afford it amalgamate their copper plates with gold. Those which cannot use silver, and those which cannot afford either, use mercury only. This applies, of course, to new plates. A plate amalgamated with gold will work quicker and better, but the use or disuse of either gold or silver is a matter of economy pure and simple, as the fact of the benefits arising from using an amalgam was well known in the gold mines I visited.

MR. N. S. KEITH: I am much pleased that a contribution of mine to the metallurgy of gold has been of use to Professor Richards in the experiments which he has just described.

My article was written nearly ten years ago, while I was still in the practice of metallurgical engineering, and was a statement of

actual, not theoretical treatment of copper plates, which I had been practicing for several years with good success, and had imparted to others. Yet I must say these results astonish me, even when obtained by the improvements so successfully applied by Professor Richards.

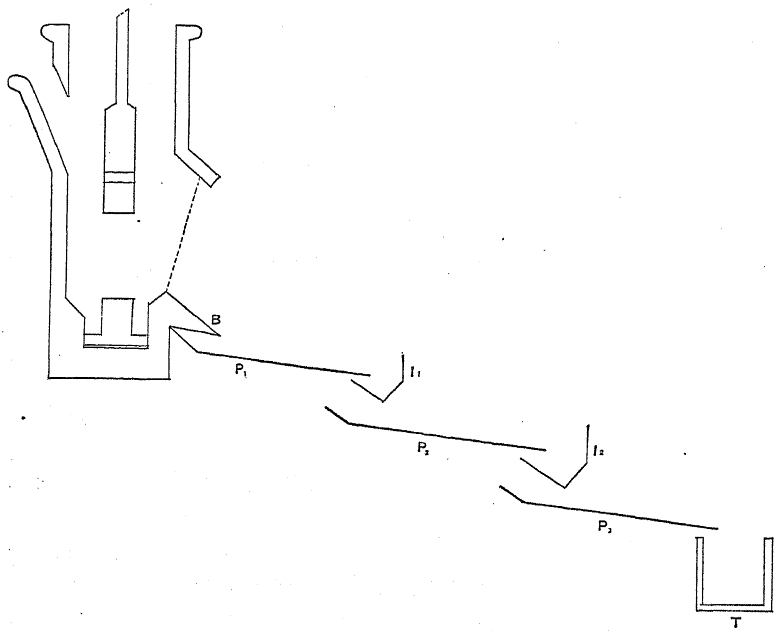
After such a lapse of time I cannot recall at the moment the full substance of the contribution referred to, but I may say that the main object was so to treat the copper plates that they were rendered highly porous by the solution of oxides and salts of copper, and the pores thus formed became the lodging-places for the amalgam. To this amalgam more amalgam readily attaches itself, and a comparatively unoxidizable coating is thus quickly formed, which protects the copper, and forms a surface which has more avidity for gold than a miser.

I must specially commend this manner of practically testing the value of gold rock in comparatively large quantities. It may be considered an assay of such material by the ton. In truth, I think it of more practical value than an ordinary assay, or a series of assays, because small, correct samples are so hard to get, and are so seldom selected by the mine-owner. We all know he never believes that the best working processes yield one-tenth of the gold his lot of ore contained, or that an assay of a correct sample is the true exponent of its value.

In times like the present, with a raging gold and silver mining fever, men seem to lose their senses, and the advice of sober mining engineers, supported by such practical demonstrations as these which have been brought forward by Professor Richards, will serve to quiet undue anticipation into an orderly, honest, and businesslike pursuit of profit from legitimate mining.

In answer to the question whether pure copper is not better for use, I have to say that although I have not tried it, I am inclined to the opinion that from its homogeneity it would not answer as well. The action of the chloride of ammonium and of ammonia in this process of preparing copper plates is to dissolve the oxides and carbonates of the electro-positive metals associated with the copper. This solution leaves the surface of the copper plate somewhat roughened or spongy. The mercury and amalgam, or better amalgam alone, fill these minute interstices in the purified copper, and present a new face, comparatively unoxidizable. Pure rolled copper has a surface that is smooth and impenetrable to mercury, and would probably hold only a film of amalgam without sufficient body to act efficiently.

MR. JOSHUA E. CLAYTON remarked that his experience for years went to show that with the use of battery plates from 60 to 75 per cent. of the amalgamable gold would be saved in the battery, and that most of the remainder would be caught upon three narrow plates, each 8 inches long, running across the whole width of the battery.



At the lower edge of these plates he placed a little iron trough with $\frac{1}{4}$-inch holes in it. This trough serves to deliver the sand in vertical streams on the plate next below, which is found to be the most favorable method of obtaining the amalgamation. He found that most of the gold collected around these little streams. The arrangement of plates preferred by him is shown in the diagram. P_1, P_2, P_3 represent the copper plates, 8 inches long, each. I_1, I_2 represent in section the iron troughs with holes in them. B is the battery discharge, and T is the waste launder. The drop of the streams as they come on to the plates should be $1\frac{1}{2}$ inch high. If a greater height is used it bores holes through the copper; if less, the amalgam builds up like little stalagmites. With very fine gold, difficult to amalgamate, this system of plates can of course be multiplied.

*THE MANUFACTURE OF CHARCOAL IN KILNS.**

BY T. EGLESTON, PH.D., SCHOOL OF MINES, NEW YORK CITY.

THE manufacture of charcoal in kilns was declared many years ago, after a series of experiments made in poorly constructed furnaces, to be unprofitable, and the subject is dismissed by most writers with the remark that in order to use the method economically the products of distillation, both liquid and gaseous, must be collected. It is asserted with truth that the number of kilns required for a large production of iron would be so great that it is doubtful whether any works would be justified in making the necessary outlay for their construction, but it can also be asserted with equal truth that for any such production in a given locality the use of charcoal as fuel would be equally unjustifiable. It appears to have been forgotten that there were other manufactures besides iron in which large amounts of charcoal were used, but these consumers seem to have been guided entirely by the results of the experiments of the iron men, who necessarily made the largest quantity of it. Some authors speak in a doubtful way of the quality of the charcoal produced, and a few concede that with great care good charcoal can be made in kilns, but that most of the workmen do not like kiln charcoal. This is the real secret of the opposition to this method of manufacture.

Most of these objections were written when charcoal was the most important fuel used by the iron manufacturer, and when what was a large production of iron for several furnaces was less than that produced by a single modern one, in localities where other fuels can be had. Almost all that has been written on the subject related to the use of the fuel for the manufacture of iron in the blast furnace, where very peculiar properties are required, and where the quantity produced is limited by the height of the furnace, and this in turn by the crushing resistance of the charcoal. A large production was, therefore, only possible when the best and hardest woods were used. Of all these objections there is not a single one which is not equally applicable to any method of charcoal manufacture, whether it be meilers, pits, or furnaces. Until recently, therefore, the manufacturers of charcoal iron

* Read at the Pittsburgh meeting, May, 1879.

have considered meiler or pit coal superior to kiln-made coal, but the manufacture of this latter kind of coal has been so much improved of late years that it is sometimes difficult for the advocates of meiler coal to distinguish the difference between the two. Occasionally the question of the sale of the accessory products was a factor. It is generally agreed that kilns give a better yield than meilers, but it is objected that the charcoal is not so strong for blast-furnace use. For other uses there does not seem to have ever been much question. Whether it will or will not be worth while to manufacture the accessory products will be determined by varying commercial conditions.

The question of pit or kiln coal was formerly settled by the cost of transportation. When transportation was low, kilns were used, the advantage of output being greatly in their favor, since the kiln can be burned slow or fast to make the coal of requisite density. The yield of charcoal is also greater in the kiln than in the meiler, it being from 45 to 50 bushels to the cord in the kiln, and from 30 to 35 bushels in the meiler. The amount of labor in using the kiln is also less. To counterbalance the increase in yield, the decrease in labor, the security against accident, and the celerity of the operation, the cost of transporation will have to be high.

Besides this the kiln is always under complete control, and can be examined by the burner at any time, and the exact condition of every part of it can be ascertained at every step in the process. As there is only an approximate knowledge and control of the meiler, the kiln should give the best product. The possibility of a large output is, however, the *ignis fatuus* of modern metallurgy. The kilns are "turned" so often that the charcoal is burned too rapidly, and the quality becomes poor. In the desire to have but few structures to take care of, and to have the largest possible furnace capacity with the least expense, the kilns have been made too large. Experience has shown, however, that it is more economical to use kilns of small capacity, and that the increase in the cost of the structures is more than compensated by the increase in the quality of the product; while with large kilns the diminished cost of the plant is dearly purchased at the expense of a diminished yield of coal, and by the relatively poor quality of the product.

There seems to be no doubt that, given the peculiar conditions necessary for the construction of permanent works, the charcoal manufactured in kilns is cleaner, since it is free from the sand of the cover of the meiler, and is also denser if the process is conducted slowly. The yield is at least from fifteen to twenty per cent. more, and the

expense is at least one-third less. In addition to this, as the meilers are at a distance, there is a loss of charcoal in transportation amounting to ten to fifteen per cent., so that the total gain in kiln manufacture is from twenty-five to thirty per cent. If the process was conducted as slowly, there is little doubt that the kiln coal would be equal if not superior to the meiler coal.

The only valid objection to kilns is their permanent character. In order to avoid the great cost of transporting all the wood these works must generally be situated in a country where there are streams or lakes which can be utilized for rafting or floating, or where the cost of carriage is very low, as the cost of transportation must be borne by only a small part of the material transported. But, as we have said, there are very few localities suitable for metallurgical manufactures where the interest and sinking fund on the permanent investment and the cost of transportation will overbalance the diminished cost of manufacture and the increase in the yield.*

Charcoal has been studied almost exclusively in view of the manufacture of iron in the blast furnace. It is generally conceded now that it will not be much longer used on a large scale for that purpose. Less attention is, therefore, now paid to it than its merits as a furnace fuel warrant. There are districts where no other fuel can be had, and there are processes in which it must be used so long as they exist. For making blooms direct from the ore kiln charcoal is almost universally used. As between hard and soft woods the impression among the workmen seems to be that the quality of the iron made with soft wood is better than that made with hard wood, but the consumption of the softer fuel is greater. There are besides some ores of iron of exceptional purity, but so lean that they can hardly be worked in the blast furnace. To be worked at all they must be crushed and dressed, and it is only practicable to treat the fine ore thus produced in a bloomary with charcoal. In such districts the kiln has a peculiar importance, as it gives a charcoal free from dirt,—a very important consideration, as the whole object of dressing the ores is to get rid of the silica, and if dirt were introduced in the fuel the loss of iron would not only be increased, but the benefit derived from dressing the ores very much diminished.

* I have seen the kiln abandoned, because the charcoal made was not good, in both Northern and Southern States, when the wood was cut in the summer and charged with the leaves on. Good charcoal could not be made of such wood by either the kiln or the meiler process. No wood should be cut for metallurgical charcoal before the fall of the sap or after its rise.

As there are many silver districts in the West where coke cannot be had at such a price as will allow of its being used, and where the ores are of such a nature that wood cannot be used in a reverberatory furnace, the most economical method of making charcoal is an important question.

Kilns for the manufacture of charcoal are made of almost any shape and size, determined in most cases by the fancy of the builder, or by the necessities of the shape of the ground selected. They do not differ from each other in any principle of manufacture, nor does there seem to be any appreciable difference in the quality of the fuel they produce, when the process is conducted with equal care in the different varieties; but there is a considerable difference in the yield and in the cost of the process in favor of small over large kilns. The different varieties have come into and gone out of use mainly on account of the cost of construction and of repairs. The object of a kiln is to replace the cover of a meiler by a permanent structure. Intermediate between the meiler and the kiln is the Foucauld system, the object of which is to replace the cover by a structure more or less permanent, which has all the disadvantages of both systems with no advantages peculiar to itself.

The kilns which are used may be divided into the *rectangular*, the *round*, and the *conical*, but the first two seem to be disappearing before the last, which is as readily built and much more easily managed.

All varieties of kilns are usually built of red brick, or, rarely, of brick and stone together. Occasionally refractory brick is used, but it is not necessary. The foundations are usually made of stone. There are several precautions necessary in constructing the walls. The brick should be sufficiently hard to resist the fire, and should therefore be tested before used. It is an unnecessary expense to use either second or third-quality fire-brick. As the pyroligneous acid which results from the distillation of the wood attacks lime mortar, it is best to lay up the brick with fire-clay mortar, to which a little salt has been added; sometimes loam mixed with coal-tar, to which a little salt is also added, is used. As the principal office of this mortar is to fill the joints, special care must be taken in laying the bricks that every joint is broken, and frequent headers put in to tie the bricks together. It is especially necessary that all the joints should be carefully filled, as any small open spaces would admit air, and would materially decrease the yield of the kiln. The floor of the kiln was formerly made of two rows of brick set edgewise and

carefully laid, but latterly it is found to be best made of clay; any material, however, that will pack hard may be used. It must be well beaten down with paving mauls. The centre must be about six inches higher than the sides, which are brought up to the bottom of the lower vents. Most kilns are carefully pointed, and are then painted on the outside with a wash of clay suspended in water, and covered with a coating of coal-tar, which makes them waterproof, and does not require to be renewed for several years.

The kilns were formerly roofed over with rough boards to protect the masonry from the weather, but as no special advantage was found to result from so doing, since of late years they have been made waterproof, the practice has been discontinued.

The wood used is cut about one and a fifth meters long. The diameter is not considered of much importance except in so far as it is desirable to have it as nearly uniform as possible. When most of the wood is small, and only a small part of it is large, the large pieces are usually split, to make it pack well.

It has been found most satisfactory to have three rows of vents around the kiln, which should be provided with a cast-iron frame reaching to the inside of the furnace. The vents near the ground are generally five inches high, the size of two bricks, and four inches wide, the width of one, and the holes are closed by inserting one or two bricks in them. They are usually the size of one brick and larger on the outside than on the inside. These holes are usually from 0.45 m. to 0.60 m. apart vertically, and from 0.80 m. to 0.90 m. apart horizontally. The lower vents start on the second row of brickwork above the foundation, and are placed on the level with the floor, so that the fire can draw to the bottom. There is sometimes an additional opening near the top to allow of the rapid escape of the smoke and gas at the time of firing, which is then closed and kept closed until the kiln is discharged. This applies mostly to the best types of conical kilns. In the circular and conical ones the top charging-door is sometimes used for this purpose. Hard and soft woods are burned indifferently in the kilns. Hard-wood coal weighs more than soft, and the hard variety of charcoal is usually preferred for blast furnaces, and for such purposes there is an advantage of fully $33\frac{1}{3}$ per cent. or even more in using hard woods. For the direct process in the bloomaries soft wood charcoal is preferred. It is found that it is not usually advantageous to build kilns of over 160 to 180 cubic meters in capacity. Larger furnaces have been used and give as good a yield, but they are much more cumbersome to manage.

The largest yield got from kilns is from 50 to 60 bushels for hard wood to 50 for soft wood. The average yield, however, is about 45 bushels. In meilers two and a half to three cords of wood are required for a hundred bushels, or thirty to forty bushels to the cord. The kiln charcoal is very large, so that the loss in fine coal is very much diminished. The pieces usually come out the whole size, and sometimes the whole length of the wood.

RECTANGULAR KILNS.

The rectangular kilns were those which were formerly exclusively in use. They are generally constructed to contain from 30 to 90 cords of wood. The usual sizes are given in the table below.

	1	2	3	4
Length	50	40	40	48
Width	12	15	14	17
Height	12	15	18	18
Capacity in cords,	55	70	75	90

1 and 2. Used in New England. 3. Type of those used in Mexico. 4. Kiln at Lauton, Mich.

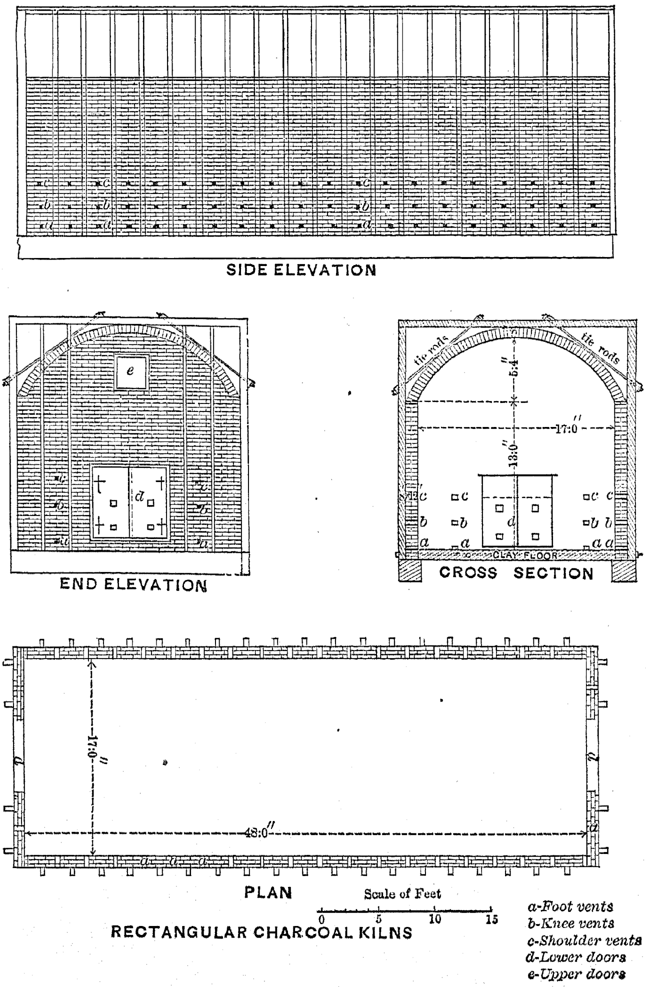
The arch is usually an arc of a circle.

A kiln of the size of No. 4, as constructed at the Michigan Central Iron Works, with a good burn will yield 4000 bushels of charcoal.

The vertical walls in the best constructions are twelve to thirteen feet high and one and a half brick thick, containing from twenty to twenty-two brick to the cubic foot of wall. To insure sufficient strength to resist the expansion and contraction due to the heating and cooling, they should be provided with buttresses, which are one brick thick and two wide, as at Wassaic, N. Y., Fig. 3; but many of them are built without them, as at Lauton, Mich., Figs. 1 and 2. In both cases they are supported with strong braces, from three to four feet apart, made of round or hewn wood, or of cast iron, which are buried in the ground below, and are tied above and below with iron rods, as in Figs. 1, 2 and 3, the lower end passing beneath the floor of the kiln. When made of wood they are usually eight inches square or round, or sometimes four by eight placed edgewise. They are sometimes tied at the top with wooden braces of the same size, which are securely fastened by iron rods running through the corners, as in Figs. 1 and 2. When

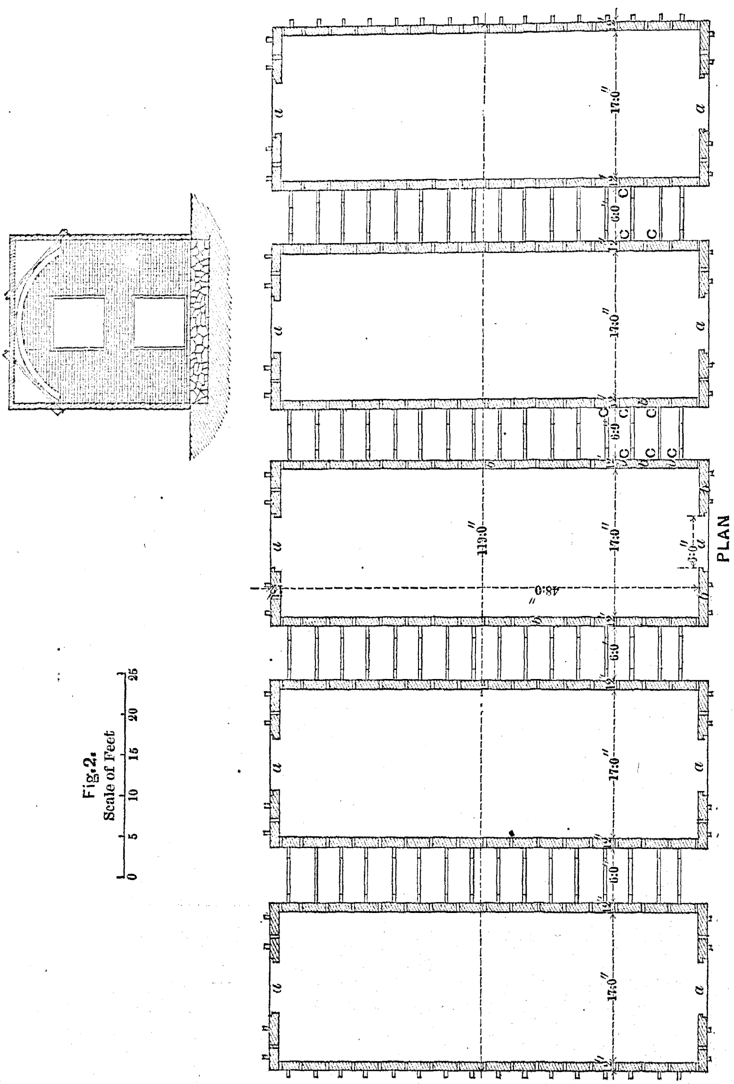
a number of kilns are built together, as at the Michigan Central Iron Works, at Lauton, Mich., shown in Fig. 2, only the end kilns are braced in this way. The intermediate ones are supported below by wooden braces, securely fastened at the bottom. The roof is

Fig. I.



always arched, is one brick (or eight inches) thick, and is laid in headers, fourteen being used in each superficial foot. Many of the kilns have in the centre a round hole, from sixteen to eighteen inches in diameter, which is closed by a cast-iron plate. It requires from

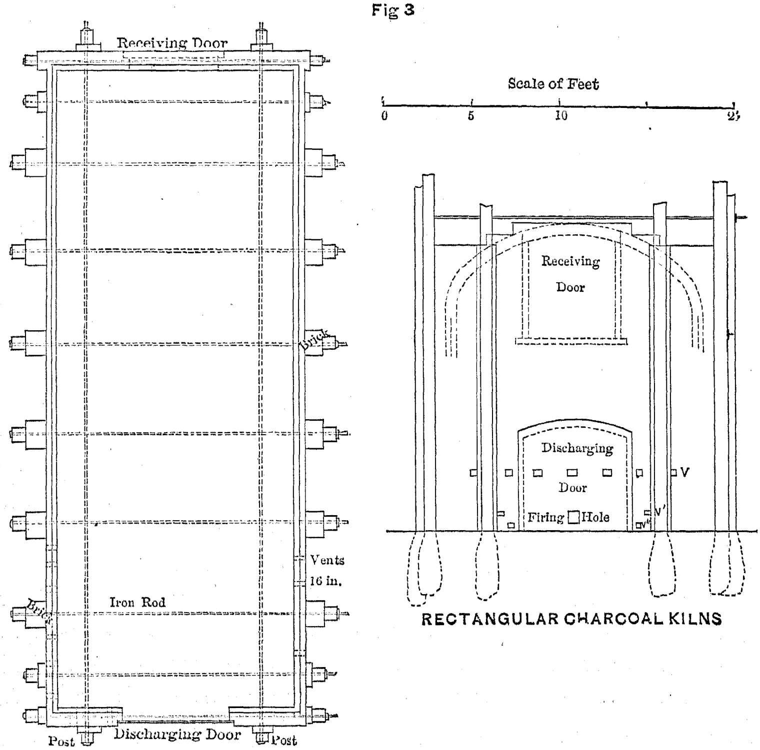
35 M. to 40 M. brick for a kiln of 45 cords, and 60 M. to 65 M. for one of 90 cords. At one end are two doors, one above, and one below (Fig. 1), both of which are used for the introduction of the



wood; at other times there is only the lower one. The discharge of the charcoal is made by the lower doors only. These doors are generally five feet square, and are supported on hinges in a cast-iron

frame. They are best made of cast iron, in two parts. Sometimes they are made of a single piece of sheet iron. The upper opening is sometimes the same size as the lower one, and sometimes about thirty inches square, and is also closed with a sheet or cast-iron door. On the sides, between the braces or buttresses, there are rows of openings, called vent-holes, regularly spaced, vertically and horizontally, for the introduction of the air necessary for combustion, and to allow

Fig 3



RECTANGULAR CHARCOAL KILNS

the products of distillation to escape. Vertically there are three or four of these, three being the most common number in this part of the country. Between the braces, horizontally, there may be only one row, where the braces are very near to each other, as shown in Figs. 1 and 2; or two rows for the lower and one for the upper, or three in each row, as in Fig. 3. The number is regulated by the way the bracing is done. When wood only is used, as in Figs. 1 and 2, there is only one, but when brick buttresses are used generally three. They are almost always placed one directly over the other. To each of these a name is given. The lower one, Fig. 1, is called the foot

vent, the middle one the knee vent, and the upper one the shoulder vent. On the ends of the kiln the vents are placed less regularly than on the sides. Sometimes there is only one row, one above the other, as in Figs. 1 and 2, sometimes one row not in line, as in Fig. 3, and sometimes there are more. From three to four vent-holes are usually made in the lower doors. These openings are usually of a size to admit a single brick. They are placed on the level of the ground and up to a height of one and two feet.

The wood is usually cut to a length of four feet, and laid flat, piled as closely as possible, so as to leave very few interstices. Stronger charcoal would probably be produced if it were placed on end, but the extra cost of labor to pile it in this way would more than compensate in expense for the extra quality. Four men, working one day, are required to fill a 45-cord kiln. The sticks are laid on the floor as closely as possible, putting the logs in the direction of the length of the kiln, and piling them in successive steps, so as to fill it up close to the arch. When the kiln is to be lighted from the top a chimney is left directly under the roof in the centre of the kiln, in which, as it is piled, dry wood and chips are placed for lighting the charge. When it is to be fired from the lower doors, a channel is made from one end of the kiln to the other, which is filled at once. The wood is piled in this way, as far as the lower door will allow, from the inside. When that is no longer possible these doors are closed, and the space that remains unfilled is filled from the outside through the upper door. When the fire is kindled the doors and the opening in the roof are carefully luted with lime mortar. If the clay used in coating the walls was used it would crack and admit air. Lime mortar must also be used in closing the vents, after the charge is ready to be extinguished.

There are several methods of lighting the kiln. In Sweden, where these kilns were first used, they were always fired from a permanent structure in the centre of the kiln, to which access could be had from the outside. In this country no permanent structure is made. When first used in New England they were lighted in one corner. The fire was then drawn along the long side, and so round the kiln on the outside of the wood, so that the heat acted directly on the walls. Such a direct exposure to the heat and pyroligneous acid rapidly destroyed the mortar in the walls, which became cracked, and after a short time the yield of the wood in charcoal was materially diminished. This method of firing was tedious and uncertain. It required from twenty to twenty-eight days to turn a sixty-five-

cord kiln, the time depending on a great variety of circumstances. Twelve to fourteen turns were the most that could be expected of a kiln in a year. The lighting is sometimes done by a chimney left in the wood in the centre of the kiln, from above, exactly as in the meiler, the fire being drawn down by the openings in the sides; this chimney may or may not connect with a channel leading to the discharging door. This method is used in the South and Southwest of the United States, and is practiced in Mexico, and is preferred when the kilns are very wide. It is considered by many to give the best results, both as to yield and quality of charcoal.

Sometimes the lighting is done by means of a channel built through the middle of the kiln, having an opening at each door. This is also filled with dry wood and shavings, which are lit from the back door. The fire is then drawn through the wood to the front door by properly manipulating the vents. When the fire has reached the front both doors are closed, as the whole kiln is then lighted. This method is called the *centre burn*. In both these last methods, as the fire is generated in the wood, the heat does not affect the walls of the kiln. The time required for these methods is not more than half that of the first method. A sixty-five-cord kiln can be easily turned twice in four weeks, which is ample time. As twenty-four turns can be made in a year, the capacity of the kiln is doubled.

In the Mexican type of furnace (No. 3, p. 378), where the lighting is done from the centre, the work is much more slowly done. It takes them four days to burn, six to cool, and four to empty, or twenty days in all, so that only eighteen turns a year are made.

The operation is easily and regularly conducted providing the walls are tight. The clear blue smoke appears about the fourth, fifth, or sixth day, when the vents must be successively closed. Four to eight days are required for cooling.

That the charge is ready to be drawn may be ascertained by taking off the cover on the top and opening a few of the vents. If after two or three hours there is no appearance of fire or of smoke the kiln may be drawn. If the kiln is opened too soon water must be pumped in at once, as the charcoal would take fire, and the men would be endangered from the carbonic acid generated, and all the men about the works would have to go to the rescue at once. A very small quantity of water is sufficient to extinguish a kiln, as the water above puts out the fire in contact with it, and what soaks through generates steam below, which extinguishes the fire in the interior. When a very dense coal is required the kiln is allowed to

die out, as it is generally thought that cooling with water impairs the value of the charcoal for blast-furnace use.

The whole art of the process consists in closing the vent-holes at the proper time. As soon as the kiln is well lighted the two lower rows are closed. The vents of the upper row are closed little by little. Those of the middle in the upper row are usually closed first; then those from which the blue smoke comes in their turn. When the vents of the first row are being closed those of the middle are opened, and so on. One man by day and one by night can easily superintend five to six such kilns. He has little to do except to draw the fire regularly down, to watch the color of the smoke so as to close the vents at the proper time, and to fill any cracks that may form. It is always safer to allow the kiln to remain a longer than a shorter time to cool. If the kiln is properly extinguished four men can easily empty it in one day. On the floor there will usually be found some badly burned wood; these brands are either put back in the next charge or used for special purposes about the works. If occasion required they might be used mixed in small quantities with the charcoal.

The rectangular kiln is going out of favor, for the reason that there is so much space heated, with such thin walls, that the number of braces has to be very large to secure a tight kiln, and when once cracked it is almost impossible to repair it so as to make it tight. The kiln is most easily managed when large, but it has been found not to be economical to use very large kilns, and as this shape of kiln is least adapted for working on small quantities it is rapidly going out of use. The tendency is now to use smaller ones, so constructed as to require the least amount of repairs. There are, however, many charcoal-burners who claim that with this kiln the fire can be better controlled while burning, and that it is easier to fill and discharge, so that it is still used to some extent in the West and Northwest States. It is always better to conduct the process slowly; more is gained in the improvement in the quality of the charcoal than is lost in time. There is of course a limit beyond which this must not be carried.

Mr. T. F. Witherbee gives the following table as the weights of the charcoal made by him in rectangular kilns at the Fletcherville furnace:

White pine,	9.800	lbs.	per bushel.	Black ash,	14.475	lbs.	per bushel.
Basswood,	10.625	"	"	White ash,	16 325	"	"
Spruce,	11.250	"	"	Beech,	17.025	"	"
Poplar,	12.275	"	"	Yellow birch,	18.750	"	"
Hemlock,	12.850	"	"	Sugar maple,	18.950	"	"

The repairs to these furnaces are exceedingly small, if made as they occur without delay, and consist mostly in filling up any cracks which may be produced in the masonry, of renewing the posts used for braces, which if properly prepared will last almost as long as the furnace, and of replacing the pieces of the coating which fall. In some places the furnace is whitewashed after every operation.

When a large quantity of charcoal is required it is generally best to construct the kilns together, as shown in Fig. 2, and in order to facilitate discharging to have a lower door at each end. They should be arranged so that the level on which the wood is stored will be about on a level with the lower part of the upper door. They should be connected with the bank by a bridge, leaving a space of from twelve to fifteen feet between the kiln and the bank below. This makes an economical and convenient disposition both for charging and discharging the kiln.

STATISTICS OF RECTANGULAR FURNACES.

CHARGING.			BURNING.			COOLING.	DISCHARGING.			Per cent. of brands.
Time.	Days' work.		Time.	Days' work.		Time.	Time.	Days' work.		
Days of 10 hrs.	Men.	Horses.	Days of 24 hrs.	Men.	Horses.	Days of 24 hrs.	Days of 10 hrs.	Men.	Horses.	
3	3	0	8	1	0	8	2	3	0	6 to 10

	New England.	Mexico.
Length, in feet,	40	40
Width, " "	17	14
Height, " "	23	18
Capacity in cords,	90	75
Yield in bushels to the cord,	50	50
Days for filling,	3	4
Days for burning,	6	6
Days for cooling and discharging,	6	8
Number of thousand brick,	60	50
Size of vent-holes,	2½ x 4 inches.	

	Wassaic.	Barnum & Richardson.
Capacity in cords,	70	90
Length in feet,	40	48
Height "	17	18
Width "	14	17
Size of discharging door,	7 x 7 feet.	5 x 5 feet.
Size of charging door,	5 x 7 feet.	28 x 30 inches.
Number of thousand brick,	30	36
Number of cubic feet of wood,	312	333
Weight of iron braces,	450 pounds.	1200 pounds.
Square feet of sheet iron for doors,	60	90

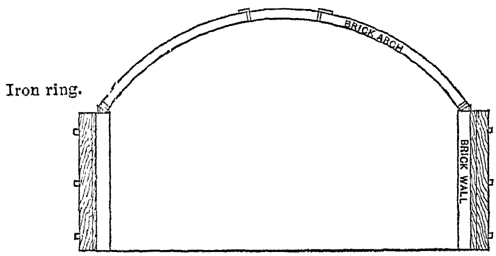
TIME OF BURNING.

	Side burn.	Top burn.	Centre burn.
Days of filling	4	4	4
Days of burning	14	6	5
Days of cooling	6	6	3
Days of discharging	4	4	2
Total	28	20	14
Number of turns a year	13	18	24

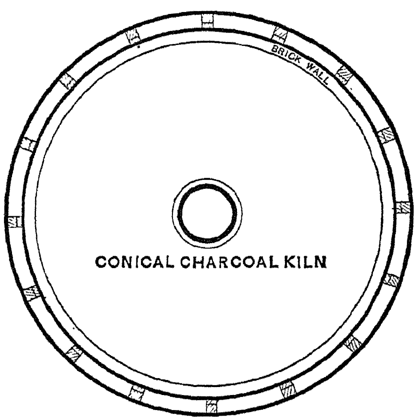
ROUND KILNS.

The round kilns, Figs. 4 and 5, differ from the rectangular ones only by their shape. Fig. 4 shows the form usually employed among the bloomary forges of Northern New York; Fig. 5 is that employed

Fig 4.



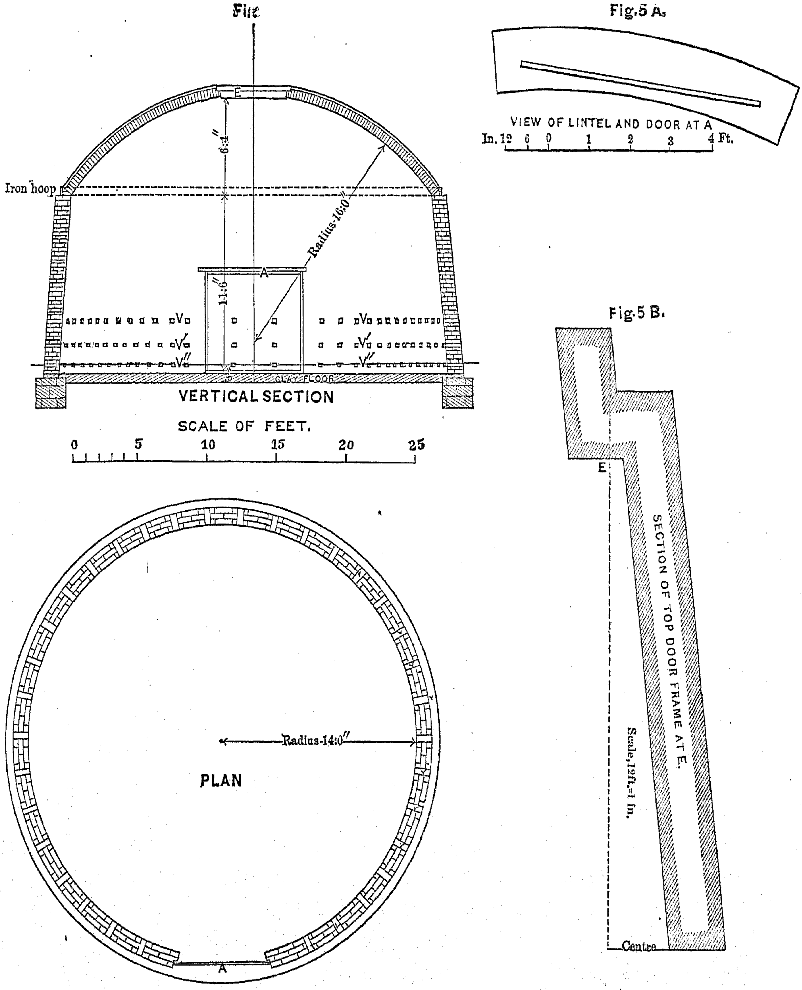
Scale, 4 ft. = ¼ in.



for blast-furnace fuel in Vermont; Fig. 4 is built with vertical walls and is braced with wooden posts and iron straps. Such kilns can

be seen near Roger's Rock, on Lake George, and at Black Brooke, Essex County, N. Y.; Fig. 5 is built with battered walls, which are 28 feet at the base and 26 feet at the spring of the arch, and has no bracing. In all other respects the kilns are similar. The arches of both these kilns are supported by iron rings at their base. Fig.

Fig.



CIRCULAR CHARCOAL KILN

5 A shows the lintel of the discharging door; Fig. 5 B shows the shape of the cast-iron door E at the top of the kiln. They have three rows of vents, one at the base, with six courses of brick between each of the others. These vents are 2 x 4 inches and 18 inches apart.

They are usually 28 to 30 feet in diameter at the base, and 26 to 28 feet at the spring of the arch. The vertical walls are 11 to 12 feet to the spring of the arch and 1 foot thick. The arch is 8 inches thick and is laid in headers. Such a kiln will have about 300 cast-iron vents, which weigh about eight pounds each. Thus a capacity of from 40 to 45 cords requires more precautions in building when the walls are not battered, and besides the usual braces, must be hooped with strong iron bands. The doors are made of No. 10 sheet iron. The top opening is a cast-iron ring, which together with the lintel of the door weighs about 1500 pounds. The wrought-iron bands around the kiln weigh about the same. They require about 36 M. brick. These kilns are usually built against a bank, so that the top can be reached from it.

The filling is commenced by placing pieces about 0.15 m. in diameter radially, leaving interstices for air-channels; above this about 0.60 m. of dry wood and brands are placed, leaving a space in the centre about 1.20 m. in diameter for a chimney, which is carried to the top of the kiln. The wood is piled horizontally, and packed as closely as possible. When no wood can be put in at the discharging door it is closed and the wood is thrown in from above, and piled as before. Shavings and light wood are then put in the chimney so that they will not pack. In some cases the chimney is filled as the piling proceeds.

With ordinary facilities it takes four to five men one day to fill a forty-five-cord kiln with wood four feet long. Where butt ends of logs are used it takes twelve hours to fill a thirty-five-cord kiln; but these can be tumbled in, and do not require much handling.

The kiln is fired with a long torch through the door at the bottom, in the space left by the skids. At the time of firing the vents are all open, but as soon as the lighting is finished the two lower rows as well as all the other openings are closed with loose brick. It takes about ten to twelve days to burn a fifty-cord kiln, and about six to seven to burn a thirty-five-cord one. A thick white smoke comes from the upper vents for about four days, during which time the water is driven out of the wood as steam. It then becomes yellowish, and continues so from one to four days, varying with the size of the kiln and the weather. It then becomes blue, and is not allowed to remain so for more than twelve hours, for the blue smoke is an indication that the kiln is very hot and that the firing is nearly finished.

It takes from five to six days after all the vents have been closed

to cool a kiln of from thirty-five to fifty cords. The covers of the doors are of sheet iron, and so long as there is any heat in the kiln it will be felt on these doors. As soon as these are cold, the kiln is cool enough to draw. It is generally the practice to let the fire die out. It is not usual to hasten the cooling by throwing in water, as it impairs the value of the charcoal for blast-furnace use. When used for other purposes, however, from eight to ten barrels of water are sometimes thrown in at the top after the kiln has been closed for three days. It will take four men about a day to draw a kiln of fifty cords.

STATISTICS OF ROUND FURNACES.

CHARGING.			BURNING.			COOLING.	DISCHARGING.			Per cent. of brands.
Time.	Days' work.		Time.	Days' work.		Time.	Time.	Days' work.		
Days of 10 hrs.	Men.	Horses.	Days of 24 hrs.	Men.	Horses.	Days of 24 hrs.	Days of 10 hrs.	Men.	Horses.	
7	1	0	12	1	0	6	4	3	0	10

Diameter, ...	28 feet at base, 26 feet at top.
Height to spring of arch, ...	12 feet
Capacity in cords, ...	50
Number of thousand brick, ...	31
Number of vents, ...	·175
Pounds of cast iron, ...	1500
Pounds of wrought iron, ...	1500

The chief objection to the rectangular and round kilns is their want of stability. The rectangular kilns require to be braced with wood and iron, while the circular kilns, which offer no advantages over the rectangular, must in addition be hooped with iron. This makes this kind of structure expensive in the first place. The less expensive structure, Fig. 5, does not last much longer. The contraction and expansion which the bracing is intended to prevent does take place, and after a time there will be numberless air-channels in the shape of cracks in the walls, which cannot be effectively closed, which will both increase the difficulty of management and decrease the yield in charcoal.

Kilns of this shape are being rapidly given up. It has been found much more economical to construct and to manage small conical kilns than large round ones, and these conical kilns are gradually taking the place of the large ones of other shapes.

CONICAL KILNS.

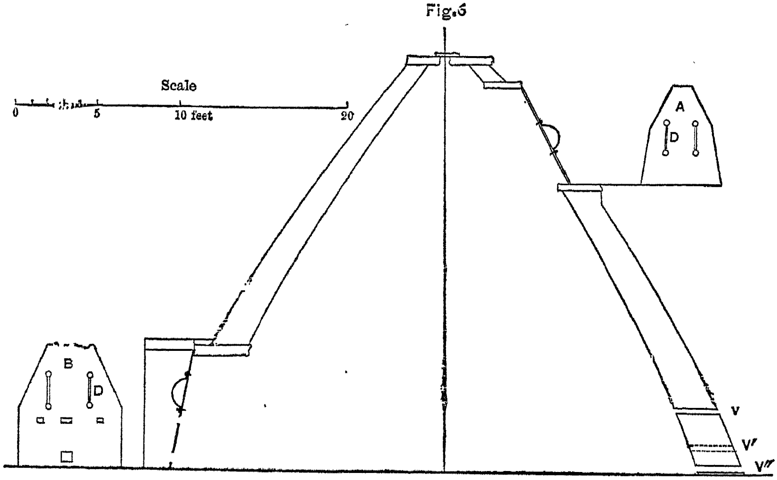
The conical kilns are generally smaller than either of the other varieties. They are usually from twenty to twenty-five feet high, and from twenty-five to thirty feet in diameter, and are intended for twenty-five to forty-five cords of wood. They are constructed in such a way as to require no bracing of any kind. They are often built into the side of a bank, a part of the earth of which is removed so as to make a charging-door near the top on a level with the ground; or they may be built on a plain, in which case there is no upper door, but only a charging-hole in the top, which is reached by a ladder in order to close it.

The usual dimensions of these kilns are:

	Diameter at base. Feet.	Height. Feet.	Capacity. Cords.
American Fork Cañon, Utah	26	20	25
Norton's Iron Works, Plattsburg, N. Y.	30	20	35
Wassaic, N. Y.	30	23	40
Readsboro', Vermont	28.6	28	45

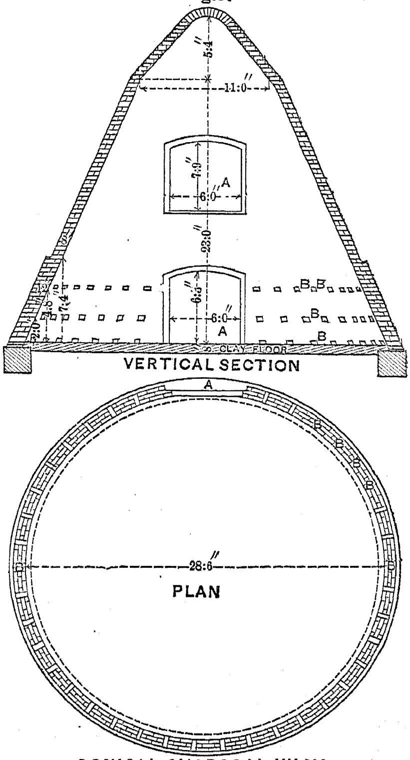
There are three types of this kind of kiln, shown in Figs. 6, 7 and 8. 1. That at Readsboro', Fig. 7, in which the top of the cone is at a different angle from the bottom; there are two doors of the same size for charging. 2. That at Wassaic, Dutchess County,

Fig. 6



N. Y., Fig. 6, in which the cone has but one angle; there are two charging doors of slightly different size, and a hole in the top of the kiln to be used in firing. 3. That at Plattsburg, No. 8, in which the conical form is the same throughout; there are two charging doors, one below, and the other in the top of the kiln. This last form is

Fig. 7.



CONICAL CHARCOAL KILNS

Scale of Feet

0 5 10 15

on the whole the best of all, being the simplest in construction and easiest to manage.

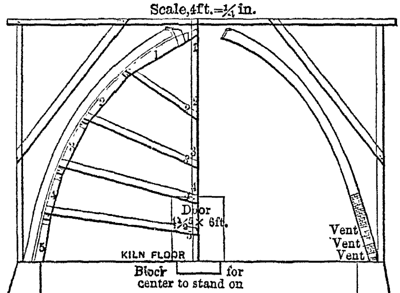
At Readsboro', Vt., Fig. 7, the walls are 12 inches thick at the bottom, but at a height of 7 feet 4 inches they diminish to 8 inches. From this height upward all the bricks are laid in headers. At the height of 23 feet the kiln is capped by another cone, which is 5 feet 4 inches high, and of much flatter angle. The details of construction are different in different localities, but the principle of all

the kilns is the same. At Wassaic, Fig. 6, the walls are made 30 inches thick at the bottom, diminishing gradually to 14 inches at the top, which is made flat for a cast-iron vent-hole. On one side a doorway, 7 feet high and 6 feet wide, is built out, into which a sheet-iron door fits against flanges made for the purpose. On the line of the American Fork Cañon Railroad the bottom walls are carried up vertically on the outside to a height of 5 feet, where there is an offset. At the bottom the walls are 18 inches thick; at the offset they diminish to 1 foot in thickness. The interior is, however, conical throughout. The foundations are of the same thickness as the lower walls, and, as they are in earth, are 10 feet in depth.

At Norton's Iron Works, near Plattsburg, N. Y., Fig. 8, the

Fig 8

Scale, 4ft.=¼ in.



CONICAL CHARCOAL KILN

wall is built with a batter of 3 inches to the foot up to 6 feet. At this point, the height of the kiln being determined as 20 feet, a perpendicular is raised, and somewhere on this line a centre is found from which an arc of a circle will meet the flange of the charging hole at the top, which is made of a cast-iron ring 4 feet in diameter and 8 inches deep, projecting 6 inches over the top. This makes the wall a little thinner at the top than the bottom. The flange of the ring is normal to the curve of the masonry, which is generally built of red brick. (Fig. 8 shows the way this kiln is constructed.) Sometimes the kilns are made of stone, which need only be strong enough to resist the low heat of the operation. Constructed in this way the kiln requires no braces of any kind. When the kiln is built against the side of a hill, the upper charging door is generally made of the same size as the lower one, but in Utah it is only 4 feet square. These doors usually have iron frames. Sometimes the upper part of the door-frame is directly under the top of the cone, as at

Wassaic, Fig. 6, and in Utah; the lower one is then on a level with the bottom of the excavation made in the side of the hill. Sometimes the upper door is much lower, as at Readsboro', Fig. 6, and is connected with the woodyard by means of a bridge. At Plattsburg the kilns are built on a plain and the top opening is then reached by ladder. The discharge door is square, and has a cast-iron frame 6 feet high and 4 feet 6 inches wide. It is best to put in at the top of this door-frame a rod $\frac{3}{4}$ of an inch in diameter, to prevent the tearing of the wall by expansion. The door itself may be in one or two pieces.

It takes 33 M. brick to construct the Plattsburg kiln. About 40 M. brick to build the one at Readsboro'. The floor of the kiln is very nearly level, and on the sides comes up to the bottom of the lower tier of vents. It is usually 4 inches higher in the centre than at the sides. It is generally made of clay 8 to 12 inches thick, and must be well beaten with mauls and very carefully tempered. The kiln has three rows of vent-holes, which usually commence on the level ground. They start on the second row of brickwork above the foundation; the floor or hearth is brought up to this level. These vents are from 2 feet 6 inches to 3 feet from centre to centre. In some kilns the vents are arranged in quincunx, so that the upper and lower rows have 24 vents, the middle one 26. In others they are one over the other, sometimes as many as 40, and the same number in each row. In most kilns these openings are at equal distances apart, which is usually 18 to 20 inches. At Wassaic, Fig. 6, they are not spaced equally. The lower ones are generally on a level with the ground. They are five inches, or the height of two bricks, high, and 4 inches, the width of one brick, wide. The vents should be made of cast iron, set into the masonry. They are made tapering very slightly, largest on the outside.

It is always wise to use cast-iron vents. If no protection to the brick is used the pyroligneous acid attacks the mortar, which falls first around the brick and then above it, and in this way cracks of considerable size may be formed in the mortar around the bricks which it is almost impossible to fill. The additional cost of the cast-iron vent-hole is about $75 to the kiln, but it will save more than this in the cost of repairs in a few years. The cast-iron effectually prevents the action of the pyroligneous acid on the mortar. In the best kilns there are three to four bricks between each vent.

Each kiln is usually constructed to hold 35 cords of wood, solid measure. This is intended to yield 1750 bushels, which is 50 bushels to the cord. Either soft or hard wood can be burned; the former

is generally used in the manufacture of blooms, and is preferred by many workmen.

To charge the kiln, skids two inches in diameter are laid 3 feet apart, keeping the ends up to the outside. They must be placed in the radii of the circle, for if the smoke is at all confined it will cause an explosion. The wood used is cut 4 feet 8 inches long, and is of all sizes from four inches up to two feet in diameter.

A fireplace, or chimney, four feet square, is made in the centre of the kiln, and carefully preserved in the filling to the top. An air-channel is made from it to the opening in the lower door. This is filled with shavings, brands, and lightwood, thrown in loose, which when the kiln is lighted is fired with a long pole from the door.

At the Norton Iron Works at Plattsburg, N. Y., the charcoal for the blast furnace and for knobbling fires is made of slabs, butt ends of logs, and flood-wood. This is brought to the kiln in a hand-cart that holds about half a cord. The bottom of the kiln is prepared as usual. Butt ends of logs are piled just above the top vents, slabs are put in up to six feet, then flood-wood about four feet long is placed, until it is no longer convenient to handle it; the filling is finished with blocks or butt ends of logs.

It takes four men about twelve hours to fill the kiln. When the kiln is ready to light, the upper door is closed and luted, but the lower one is left open until the kiln is lighted. Bricks are placed loosely in all the vent-holes; sufficient air comes in around them to light the wood. The fire is lighted with a torch through the channel prepared for the purpose. Four vents on each side of the door are then closed and luted; the others are closed as it is necessary, with a loose brick. The firing is always done at night. At the end of thirty-six hours a little air is let in by the vents near the door. The operation is conducted by means of the vents, which are carefully watched. The smoke will be white from three to four days. It will then become yellow in from 12 to 36 hours, and then will be blue for about 12 hours. When the smoke is blue at the top vent, it is allowed to remain open for twelve hours, and is then closed. In another twelve hours the second row will blue up, when that is closed. The bricks are finally drawn out of the lower vents; the fire then comes to the bottom, and when ready they are closed.

If the smoke is pale and in small quantity at the commencement the vents must be opened. When it is burning well the smoke is quite like steam. It takes from 6 to 8 days to burn a 35-cord kiln. All the openings in the kiln are then carefully closed and luted.

After it has stood $2\frac{1}{2}$ to 3 days, 8 to 10 barrels of water are thrown into the top opening, or through the vents, and the next day the charcoal can generally be drawn. It will take five days to cool without watering, and three if water is used. It is necessary before drawing that the fire should be perfectly extinguished. To be certain that the fire is quite out, the charging door or vent-holes are sometimes opened for an hour or two. If there is no increase in the temperature of the air the kiln may be discharged.

The yield of the kilns at Norton's Works is often as high as 60 bushels per cord for hard, and 50 for soft wood. The average is about 50 bushels.

It takes four men and two horses one day to fill the kiln. One man is required in the kiln, and three to draw. Two men can empty it in a day. The coal at Norton's Works is carried from the kiln to a shed only about thirty to forty feet distant, which is forty feet deep, having a peaked roof. The building is 275 feet long.

There are thirteen kilns here, and four at twelve miles distant, built of firestone, which hold fifty cords each. It requires eight men for the thirteen kilns. These thirteen kilns can make twenty-two turns a year. Working so fast makes it difficult to burn the charcoal thoroughly, so that they usually run only eighteen kilns a year. It requires thirteen days to fill, burn, and empty a 35-cord kiln. It takes one day to fill, and one to empty it.

In Utah the charcoal is packed in bags, containing two to three bushels, which are hooked up under an opening eighteen inches square in a table three feet high. This table is four feet square on the top. The charcoal is thrown on to the table and pushed into the bags. Six men will empty a kiln, sack the charcoal, and put it on the cars in one day. The wood is mostly poplar, which is cut dead. The charcoal is very light, and so much was lost by this method that it is now proposed to ship in cars constructed for the purpose without packing. It takes nine to ten days to burn a kiln. At the end of five days all the vents are closed, and they no longer watch it. The fire is extinguished with water; it is not allowed to die entirely. The kilns are whitewashed after every operation. Poplar costs \$3 per cord. Pine costs \$3.75 per cord. Six men do all the work. The loss in fine coal is very large. The charcoal made from those very soft and often dead woods is very light, so that it crushes by its own weight, and bears transportation very badly. The use of such woods is, however, partially justified by the fact that the

coke for the lead furnaces costs in this locality $45 per ton. Thirty-five cords of wood make 1000 to 1200 bushels, of 2680 cubic inches each, of charcoal, or about 45 bushels to the cord. In 1875 the contract was made to deliver at American Fork at 18 cents a bushel.

It seems to be very generally conceded in the Eastern States that the conical kilns holding from 25 to 35 cords are the most profitable. They are less expensive in construction, more easily filled, cheaper to manage, give a better yield, and can be turned more frequently than any of the other varieties of kilns.

STATISTICS OF CONICAL FURNACES.

CHARGING.			BURNING.			COOLING.	DISCHARGING.			Per cent. of brands.
Time.	Days' work.		Time.	Days' work.		Time.	Time.	Days' work.		
Days of 10 hrs.	Men.	Horses.	Days of 24 hrs.	Men.	Horses.	Days of 24 hrs.	Days of 10 hrs.	Men.	Horses.	
1	6	1	8	1	0	3	1	4	2	5

Diameter or length,	28 feet inside in the bottom	
Height to spring of arch,	19½ feet	
Capacity in cords,	35	
Number of thousand brick,	33	
Number of vents,	94	
Weight of cast-iron vent-holes,	14 lbs. each	
Pounds of cast iron,	3000 lbs.	
" wrought iron,	1000 lbs.	
Cost per hundred bushels,		$ 0 85
Foundation 50 P. of stone, 3 to 4 feet are required, cost,		100 00
Bricks, $5.50 per M., Laying in wall, $4.50 per M.,		320 00
Cast iron for frames, vent-holes, etc., 1 ton,		75 00
Total cost of furnace,		$495 85

At Plattsburg, in 1879, it cost $7.50, on Lake George $7, and in a few localities in Vermont $6 per thousand bushels to fill, burn, and empty; the average will be from six to seven cents per bushel.

The per cent. of brands in a well-burned kiln will be from one cord in 17.5 cords to one in 18.5.

The charging requires 1 day for 4 men for 35 cords, and 1 day for 5 and 6 men for 50 cords. Two horses are used for the 35-cord

kiln at Plattsburg, but the whole work is often done by men with wood barrows.

	50 cords.	35 cords.
Days charging,	1	1
Men employed charging,	5 to 6	4
Horses used,	(barrows)	2
Number of days burning,	10	6 to 8
Number of men employed burning,	1	1
Number of days discharging,	1	1
Number of men "	3 to 4	2
Number of horses,	2	2
Yield of wood in bushels,	50	50
Number of bushels of charcoal to cord of wood,	50	
Weight of bushel,	20 lbs.	soft, 11 to 14; hard, 15 to 19

The cost of these kilns will vary with the locality, depending on the local cost of the materials used. It will cost about $500 to build a conical kiln of from 35 to 50 cords in Plattsburg, and about $600 in Michigan, with brick at $17.50 per thousand.

There seems to be no doubt that the Plattsburg kiln, with iron vent-holes, is the best type of all the kilns. If properly built it lasts a long time without repairs of any kind except an occasional replacing of the clay-tar wash on the outside. At Plattsburg anything in the shape of wood is made into charcoal by it, and while it is not generally advisable to use drift and refuse woods in current manufactures, it can be done if necessary. Such kilns as this can be built by almost any man. They are much easier to take care of than meilers, and in the remote districts, where charcoal is now used as a metallurgical fuel, present every advantage of economy of construction and management, as well as maximum of yield.

In conclusion, I beg to express my thanks to Mr. E. Gridley, of Wassaic, to Mr. J. A. McArthur, of Saulsbury, Ct., and to Mr. J. H. Totman, of Plattsburg, for assistance in preparing this paper.

*THE LAW OF FATIGUE AND REFRESHMENT OF METALS.**

BY T. EGLESTON, PH.D., SCHOOL OF MINES, NEW YORK CITY.

FOR several years I have been engaged in studying the behavior of iron and steel under varying conditions of tension and compression, as well as of shock and abrasion. Some of these observations have been communicated to the Institute at various times. Within the past year I have been looking at the various observations that I have made as a whole, and have been led to the conclusion that there is a regular law of fatigue and refreshment of metals, and that the change which produces either the fatigue or refreshment is a chemical one, which is, however, in almost all cases accompanied by physical and molecular changes at the same time. That metals under conditions that were more or less well understood became worthless, has been known for many years. It has been known to blacksmiths that iron and steel, when improperly treated in their fires, become "burned," as they call it, and could no longer be used, and some of them have had a scrap heap into which such material was thrown. A few of them knew that after a period more or less long the iron or steel recovered some of its original properties and could, with many precautions, again be used, but that these phenomena followed any law has not to my knowledge been announced.

My attention was first attracted to the varying conditions under which iron and steel rails break in service. It was not long before I became convinced that there was a general law applicable to all metals which allowed of their being used with safety within certain limits, and caused their rupture when the use extended beyond these limits, and this law I have called the law of fatigue of metals.

When we observe the causes of deterioration in iron and steel rails, we find that they are principally shocks, either slight and frequent, like those produced by the passing of a train over a road-bed in good order, which produces a vibration sensible to the ear laid on the rail, for very long distances, but distinctly audible for only a short one; or from sudden and heavy blows, like the descent of the locomotive and each car successively from a high to a low rail.

* Read at the Montreal Meeting, September, 1879.

Intermediate between these in effect is the constant bending backwards and forwards of the rail when the train passes over loose and low ties. Acting at the same time with all these is the cold rolling of the iron and steel, which is a very important agent in changing the structure of the rail under heavy traffic, and which also, as I showed at the Baltimore meeting,* produces a cold flow of the metal. The fractures which are produced by any or all of these causes are accompanied in the iron and steel by such a very decided change in the physical appearance of the metal, that in many cases experts have been led to pronounce the iron of unmistakably bad quality. Such was the case with the irons used in the armament of Fort Delaware, which were shattered almost like glass by the projectiles, weighing about 400 pounds, fired at point-blank range, and of the rail mentioned at the St. Louis meeting;† yet both of these materials, when examined afterward, showed a much greater strength than the contract required, and were nearly equal to the very best materials known. The same observations were made on the change in the irons broken by tensile or transverse strains. This led me to study the effect of shock. Having eight sections of steel rails to examine, some of which had borne a traffic of nearly one hundred million tons, and others which were planed out at my request from rails which had never seen service, I made careful etchings of all of them, and then with both light and heavy blows struck with punches the names of the makers of the rails and their weight into the surface of the rail filed smooth for the purpose. Every trace of the letters was then filed out and the rail end again etched, when every letter became plainly visible. These were photographed, and again filed and re-etched, and the operation again repeated, until a faint blotch, entirely illegible, was the only visible trace of the effect of the punching. These same effects are produced in the wearing of the rail in different intensity, and show both the effect of the cold rolling and of the shock of the passage of the trains very distinctly. The change produced is a chemical one, and is at the same time accompanied by a change in the size, color, and surface of the grain of the iron or steel, and when this surface becomes distinct enough over a considerable surface the faces of the crystals slip on each other, and the piece separates, or in other words, breaks in the direction in which the strain is applied. All these effects are most distinctly visible on the hook from the Brooklyn bridge,‡ mentioned at the Baltimore meeting, and on the

* Transactions, vol. vii, p. 373.

† Transactions, vol. iii, p. 63.

‡ Transactions, vol. vii, p. 375.

irons from Fort Delaware, which were afterwards used as tests to ascertain the quality of the iron, and then etched to study the effect of the strain. In both cases nearly fifteen millimeters of the material had to be filed away in order to get the smooth surface for examination by etching, but the effect of the strain was very marked.

I have definitely ascertained that when metals get into this condition from too great strain, produced in any way, or when they approach it so as to be near their limit of elasticity, that they may be brought back to their original condition either by rest, for a time more or less prolonged, or by heat, which may be applied either slowly or rapidly as may be most convenient; and this fact I have called the law of refreshment.

A very good illustration of the condition of fatigue is that of the strain produced in casting metals in large quantities. Cast iron guns which were considered so bad that it was useless to test them, have, after years of rest, proved equal to the severest tests. It is equally true of other strain, where the metal has been carried very near to the point of rupture, or even beyond it. Heat applied up to a dull red has also the property of restoring the material. These collective phenomena I have called the law of fatigue and refreshment of metals. As a law I believe the announcement is new; that the change producing fatigue is a chemical one is also new. Isolated facts about the behavior of metals under strain, etc., have been known, but that they followed a law, and that rest was necessary to the conservation of the mechanical life of a metal, is, I believe, new. That the phenomena of rupture in iron and steel was accompanied by a chemical change and a whitening of the metal is, I believe, also new. These phenomena are constant in all the metals which I have been able to study. They occur in the zinc of organ pipes, which become brittle after a time from vibration. I have seen the finger, with slight pressure, put directly through a piece of old organ-pipe, pushing the metal to one side, and leaving it broken on the sides exactly as paper would tear when the finger is pushed through it. Unfortunately the piece was regarded as a curiosity, and I was unable to get any of the metal to experiment on. The only apparent exception to this law of fatigue was recently discovered in Freiburg. Some rings and disks of tin which had been shut in an open space between two walls were found to be reduced almost to powder while they still retained their shape, and they were brought each to its original condition by the application of a gentle heat. The pieces are supposed to have been completely at

rest. From a description of the conditions under which they were found, it would seem to me that they must have been in a state of almost continuous vibration for many scores of years, as they were found high up in the steeple of a church, on the sill of a window, in a cavity supposed to have been made when the window was built up about two hundred years ago. If the condition of vibration could be proven this would cease to be an exception.

I am now having a machine built, which I have contrived for the purpose, to study these physical and mechanical phenomena. Messrs. Jones & Laughlin, Miller, Metcalf & Parkin, and the Edgar Thomson Steel Works of Pittsburgh have prepared a series of samples for me, but until quite recently I have had the greatest difficulty in getting the necessary material to work on. I hope before many months, if I can get this material in all metals, to present to the Institute an extended series of observations and tests on this subject, which has so many industrial applications to machinery, and to all cases where metals are used for structural purposes.

DISCUSSION.

PROFESSOR SILLIMAN asked Dr. Egleston if he was prepared to state in what the supposed chemical changes in the constitution of metals consisted which resulted, according to Professor Egleston, in the fatigue of metals. A molecular rearrangement in the substance of a metal, due to physical causes only, may account adequately for many, if not for all, of the facts cited by Professor Egleston as illustrating the fatigue in question.

In alloys the tendency to crystallization under vibration, percussion, and changes of temperature, or even simply by lapse of time, is a not infrequent cause of loss of strength, and even of fracture. Many illustrations of the truth of this statement will at once occur to members present. A familiar case, which every chemist may have noticed, is that of a coil of brass wire hanging unused in the laboratory becoming brittle and worthless in no very great length of time. I have observed a case in my own experience of a trellis of brass wire strung for grapes, the strings of which were strained by screw eyes. Within a few weeks, one by one, the strings successively snapped, usually on cool nights, with a loud resonance. The cross-section of fracture in these broken wires exhibited a radial structure, resembling zinc, and no portion of the wire, after this short exposure, could be bent without fracture, although before use the

wire was tough and of good quality, as was evident from its supple twisting about itself near the eye. In these cases the metal becomes fatigued without doing any work in the first case, and not much in the second.

At the suggestion of Dr. Hunt, who made the inquiry, Professor Silliman further stated very briefly the results of his experiments made upon the so-called britannia metal, an alloy of tin hardened by antimony and copper. This alloy is largely used in the production of silver-plated ware in the United States. Cast in ingots, this alloy possesses a certain degree of resonance, but after rolling into plates, and spinning on the formers in a lathe, it is quite flat and devoid of resonance. If these unmusical pieces are carefully treated in an oil-bath at a temperature just short of the melting-point of the alloy, the original resonance is restored. This obviously is due to a molecular rearrangement in the mass, and it is accompanied by a corresponding change of density, the reheated pieces being less dense than before this treatment. This case differs from that of annealing in metals, which become softer and less resonant by heating and slow cooling after heating. Nor is it the same change by which glass quenched in hot oil is toughened, under the patents of Labastie.

Professor Silliman also suggested, by way of inquiry, to Dr. Egleston the study of the phenomena of occluded gases in iron and other metals, as possibly connected with the interesting questions he has under investigation, indicating in this connection the well-known researches of Dr. Graham and Professor W. W. Wright.

Dr. Egleston: I have made no examination of alloys, and had not proposed at present to extend the research to them. There is undoubtedly a molecular change which causes the fatigue, but I think that in iron and steel, at least, the immediate cause of this molecular change is chemical, and is a recombination of the elements contained in the metal in a different way to produce the fatigue from that which corresponds to the strength to resist. The question of occluded gases has been often in my mind as a subject of research, and I have several times arranged the details of a plan for carrying out such an investigation, but up to the present time have not been able to do so.

Dr. R. W. Raymond: If I understand Dr. Egleston's partial statement of the law he has observed, it is that a chemical change accompanies the physical change induced in metals by fatigue, stress, or shock. It is difficult to conceive what chemical change

could take place in a pure metal. Dr. Egleston's most definite statements refer to the various mixtures or alloys of iron and carbon, and the chemical change he observes is the passage of the carbon from an uncombined to a combined condition. I should hesitate to call this a phenomenon attaching to metals, since it is merely the behavior of a non-metallic substance mixed with a metal. The fact itself seems to be analogous to the well-known phenomena connected with the casting and cooling of iron, and discussed some years ago by Dr. Drown in a paper before this Institute.*

If a high temperature can cause graphitic carbon in a small portion of iron to unite with the iron in which it is enveloped, then, I think, there is no doubt that the iron surrounding that small portion might act as a "chill," removing the heat so rapidly as to prevent the resegregation of the carbon, just as we see is the effect when we cast in "chills." In the latter case we have the carbon dissolved in fluid iron. In the case of a sudden blow we have a temperature less than that of fusion, but still possibly high enough to induce the recombination of the graphitic carbon.

Dr. Egleston: In the sense that Dr. Raymond speaks there are no metals, since there are no metals sufficiently pure not to contain some non-metallic substances. Most of those used in commerce contain almost as much foreign substance as iron and steel. That heat plays an important part in the phenomena in the case of sudden shocks is proven beyond doubt by the experiments at Fort Delaware. All the iron that was fatigued by the blow was so hot immediately after it that the hand could not bear the heat. But the same phenomena occur when there is no heat, or at least none that can be detected. There is a large amount of heat produced by the cold rolling of a rail, whether in or out of the track, but how much has not, so far as I know, been determined.

Dr. A. L. Holley: Can you always find the indications of the gag used in cold-straightening rails by your test?

Dr. Egleston: Invariably.

Dr. Holley: That is very important. I have recently been talking over rail straightening with Mr. Sandberg, Mr. Price Williams, and other authorities abroad; they are generally agreeing that cold-straightening must be very much reduced by a better system of hot treatment, such as we have in the Gustin hot-curving machine, now used in most of our steel-rail mills, and in Mr. William R.

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* Transactions, vol. iii, p. 43.

Jones's machine, at the Edgar Thomson Works. There is no such machine abroad; rails on the hot bed are sometimes curved in different directions, and with short crooks, so that cold-straightening is excessive. Most of the steel-rail fractures abroad are found to occur at the gag-mark. By the application of Dr. Egleston's test perhaps we should find all fractures at the gag-mark. I learned at Krupp's works that the straightener is fined 5 marks ($1.25) for a gag-mark on the flange of the rail. Gags are made in some American works so that they cannot touch the flange.

NOTES ON THE BLAST FURNACE.

BY J. M. HARTMAN, PHILADELPHIA.

One of the most important subjects to the blast-furnace engineer is a thorough knowledge of the conditions affecting the temperature in the different portions of the furnace. All efforts to decrease the consumption of fuel and improve the working of the furnace must be based upon it, and I may, therefore, be permitted to place before you the results of my observations, and detail the conclusions to which they have led me.

Tracing the thermic conditions from below upwards we have at the bottom of a blast furnace making No. 3 iron a temperature of 2900° F., which increases slightly to a point a little below the tuyeres. In the immediate vicinity of the tuyeres the temperature is somewhat lower, owing to the entering blast; but a short distance above the tuyeres, where all the oxygen of the blast has been converted into carbonic acid, the highest temperature in the furnace must be attained. This carbonic acid is, however, almost as soon as formed, converted by the glowing coal into carbonic oxide, a process which absorbs heat, reduces the temperature, and provides the active agent for the reduction of the iron ores. In its ascent the hot carbonic oxide gradually parts with its heat, first melting the descending iron and earthy materials, which drop down to the hearth of the furnace. The limit of this zone of fusion is rather sharply defined, and the temperature above this zone suddenly decreases. In its further ascent the hot gas drives off the carbonic acid from the limestone, which causes a further absorption of heat, and finally the gas escapes at the top, at a temperature of about 250° F. if the furnace is working well. These changes of temperature I have attempted to represent graphically,

without regard to actual values, in Figure 1, on the accompanying Plate.

If, now, we reverse the direction of the investigation, and trace the thermic conditions involved in the descent of the coal, ore, and limestone, we find that the charges descending at the rate of 3 feet per hour (if the furnace is driven properly) become heated at the expense of the ascending hot gas. At a temperature of about 570° F. the ores begin to be reduced, or to lose their oxygen under the influence of the carbonic oxide. The quantity of heat absorbed in deoxidizing the ores being less than that developed by the formation of carbonic acid, an increase in temperature is the result. Further down, at a temperature of about 750°, the limestone begins to part with its carbonic acid, a somewhat higher temperature being necessary where dolomite is used. Still descending, the point is reached where the iron is melted and the earthy matters are fused together as cinder. From this fusion limit downward to a point about 3 feet above the tuyeres an atmosphere of carbonic oxide exists, which prevents the oxidation of the falling shots of iron, and reduces any fugitive pieces of ore which may have escaped the zone of fusion. From about 3 feet above the tuyeres to about 6 inches below them an atmosphere of mostly carbonic acid exists. This space is called the zone of combustion, and it is upon the extent of this region that the rapidity of the driving of the furnace, or the volume of entering blast, depends. From the hearth or bottom of the furnace to the zone of fusion the furnace is filled with glowing coal, although occasionally a stray piece of refractory ore or stone will be found here.

The heat in the hearth of a blast furnace is the result of the combustion of the fuel by the blast, to which is added, in the case of hot blast, the heat brought in with the air. It is evident that if the blast be cold a correspondingly larger amount of fuel must be employed to maintain the same amount of heat in the hearth than when hot blast is used. A temperature of blast of 800° F. is needed to ignite charcoal, of 1000° to ignite coke, and fully 1300° to ignite anthracite. The convenience and advantage of contributing to the heat of the hearth by heating the blast is now fully understood.

Running on a burden of one pound of coal to one pound of ore more heat is developed than is required. The descending stock cannot absorb the large volume of heat coming up, and consequently the furnace becomes hot to the top. This excess of heat is partly absorbed by the decomposition of some of the carbonic acid in the gas by the glowing coal at the top, carbonic oxide being formed. As this

amount of coal is lost to the furnace, it is wasted. This waste, however, acts advantageously, by causing less coal to reach the hearth, and thus hindering the make of iron high in silicon. This evil exists more widely than is generally supposed, as it is, to a certain extent, self-corrective. If the burden is increased, say two pounds of ore to one of coal, and the same volume of blast used, then the heat returned per hour to the hearth from the ore and stone will be double. This heat, in combination with a higher temperature of the blast, replaces the pound of coal which it saves, and at the same time doubles the yield of iron.

Concentration of heat at the tuyeres is one of the first aims for successful furnace work. This can only be obtained by large hearths, hot blast, heavy burden, and rapid driving. The descending stock in the furnace collects the heat from the ascending gas and carries it down to the hearth again, increasing the intensity of combustion at that point—an important matter when it is considered that the intensity of combustion in the hearth determines the grade of iron.

The descent of stock in the furnace is governed entirely by the rapidity of combustion at the tuyeres. A true test of the furnace is the number of tons of material passing through it in twenty-four hours, be it coal, or ore, and stone. If the volume is large the furnace is well proportioned, and the greater the proportion of ore in the total stock the larger will be the yield of iron if the furnace process is properly managed.

The descent of the stock in a modern furnace may be represented by the dotted lines in the interior of the furnace in Fig. 1. In the upper part the stock on the sides travels faster than that in the centre, while from the bosh downwards that in the centre travels faster.

When a furnace of this shape has been in blast six months, and has been worked up to full capacity, it will assume the shape given in Fig. 2. Starting from the tuyeres we find, at a point about 3 feet above them, that the walls are cut back, and that from there upward they are nearly vertical, until a point a short distance above the mantel is reached, when the walls are again cut back. This latter cutting away is due to the fact that the thick walls above the mantel retain the heat and allow the brick to burn away. Continuing upward we arrive at a point where the rough, fretted surface of the walls suddenly disappears, and the walls are smooth from the wear of stock. This line between the smooth and rough surface marks the limit of the zone of fusion, and its height is determined

by the volume of air entering the furnace per minute. The number of cubic feet of air entering per minute divided by five will give the cubic contents of the zone from the top of fusion limit down to the tuyeres for charcoal, divided by four will give the cubic contents for coke, and divided by three the cubic contents for anthracite. These figures have been determined by measuring the area spoken of and comparing it with the air entering per minute in a number of furnaces, and, for all practical purposes, it will be found correct.

Mr. Daniel Morgan, founder of the Pennsylvania Iron Company, wishing to determine the height of the melting-point in the furnace, took four bars of $1\frac{1}{4}$-inch square iron, placed them on different points, and lowered them through his open-top furnace until the bottom ends ought to have reached the tuyeres. After they were in one month they were pulled out, and all of them found burned off exactly 7 feet 6 inches above the tuyeres. The ends were burned square off, showing that there was a sudden change of temperature at their ends. Some time after this the men at this furnace, after running out the iron at the six o'clock cast, struck and went home. As neither persuasion nor fair offers could induce the men to resume work, Mr. Morgan finally got some laboring men to clean out the furnace, which had by that time chilled. After shovelling out for some time through the tuyeres the coal stopped descending, and on examination it was found that a straight and well-defined ceiling extended across the bosh. As they could not get it down, they went to the top and took the stock out in buckets until they got down to what appeared to be a floor. After clearing this floor off, a hole was broken through it, which showed it to be 15 inches thick. This ceiling was 7 feet 6 inches up from the tuyeres, and corresponded to the height at which the rods were burned off. In this case the zone of fusion lost its heat and the stock set firmly across the bosh.

If sufficient heat gets above the fusion limit of a blast furnace to paste the stock and yet not fuse it, this stock jams on the bosh, forming a ring which, if the stock above cannot push it down to the fusion limit, becomes permanently set and obstructs the flow of stock. If the materials in the stock passed abruptly from the solid to the fluid state as ice does to water this condition of affairs could not arise, but we may rather compare the stock to wax, which when the heat is just insufficient to melt it becomes pasty and can then be moulded and compressed. The circumference of a furnace of 20 feet bosh is 63 feet, while the circumference of this furnace at the fusion limit is about 44 feet. The stock immediately on the bosh and sliding down as a

whole is about 30 inches thick, while the balance of the stock in the central portion travels independently and much more rapidly. A compression or squeezing together of $63 - 44 = 19$ feet has to take place in the stock travelling down the slope of the bosh. If the stock is pasty it squeezes together and jams as above described. This ring or "skew-back" lodges the stock above it up to the top of the furnace (see Fig. 4). The stock descending through the middle of the furnace by its side thrust retains this lodged stock in a vertical position, making a dry wall of it. This cuts off the reducing area of the furnace and proves Cochrane's law—that the yield of a furnace shows its working area. When this lodgment or scaffold occurs, the lodged part collects heat at the bottom, which accumulates and works up through it to the top of the furnace. The stock against the walls under the bell becomes red hot, while the stock in the centre is cold. This has led to the idea that a furnace sometimes works up its walls, while the reverse is the case.

After a furnace has been in this condition for some time, the attrition of the stock and an increase of temperature sufficient to melt the skew-back causes the latter to give way, and the scaffold gradually slides downward in the furnace. While the furnace is melting up the scaffold it becomes extremely hot, because the work of reduction has been done thoroughly and carbon is stored up in the scaffold. The founder in this case increases the burden and drives the furnace. As soon as this scaffold is worked out the furnace turns on white iron, unless the founder has reserve heat in the hot blast. When a furnace in this scaffolded condition is blown down, the dry wall around the sides falls in and no trace of it can be found. When the founder gets the furnace blown down to the bosh and finds no scaffold, he refills; but in a short time the old trouble shows itself. From the foregoing explanations it will be seen that the difficulty is that he did not blow it down to this ring or skew-back.

Furnaces built the usual shape, cut out so much above the tuyeres that the engine and hot blast are not large enough to fill this space, and the blast shuffles about from side to side of the furnace, causing it to work hot first on one side and then on the other. As the heat cannot be concentrated, white iron results. The furnace must then be blown out and a new bosh put in.

The difficulties of the old form of furnace may be avoided by adopting the form shown in Figure 3, placing the bosh far enough above the fusion limit to avoid the danger of the jamming of the stock on the bosh. Any pastiness of stock will occur between vertical walls, the

heat will be concentrated, and the blast equalized. The only objection to extending the bosh so high is that it will add 10 per cent. to the height of the furnace. As the upper part is simply a hopper to hold the stock during reduction its shape will be immaterial so long as the charging apparatus can distribute the stock evenly. The mantel of this furnace is higher than any used (except that on Lucy No. 2, Pittsburgh), and is placed 2 feet below the fusion limit, which can now be ascertained closely. The well is formed of a 12-inch wall, surrounded by a water jacket extending from the mantel to below the hearth line.

Two pyrometers should be placed in the sides of the furnace above the bosh and one in the escaping flue, that the working of the furnace may be controlled in case the equilibrium is disturbed by the encroachment of one zone on another. To establish and maintain this equilibrium is the work of the future. A furnace built as above described will retain its shape and take a certain uniform volume of blast, because the contents of the zones of combustion and reduction will bear a fixed relation to one another. With reserve power in the stoves, a burden of ore can then be determined upon that needs no changing, as by varying the heat of the blast iron of any quality can be produced.

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*THE LOSSES IN COPPER DRESSING AT LAKE SUPERIOR.**

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THE native copper of Lake Superior occurs in the form of fine grains and scales, disseminated in small percentage through the copper-bearing rock; and in large and small masses, from a few pounds to several hundred tons in weight.

Formerly the bulk of the copper came from the mines of Ontonagon and Keweenaw counties, producing mass copper. At present these mines furnish less than one-tenth of the total product, and fully one-half of this is from a single mine. The masses are usually found irregularly distributed in nearly vertical fissure veins, and require much dead work in sinking, drifting, and stoping for their development. The risk and expense attending the working of these mines have made them, with few exceptions, anything but profitable.

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* Read at the Montreal meeting, September, 1880.

The mass mines yield also a certain amount of stamp rock, obtained in stoping the veins in the systematic exploration for mass copper. The amount of stamp rock so obtained is however quite small, and does not suffice to keep the stamp mills running. Indeed, it is said that one good-sized mill would treat all the vein rock mined in Keweenaw County! At the Copper Falls and Delaware mines a large amount of stamp rock has been obtained from amygdaloid beds (that of the former mine known as the "ash-bed"), and it seems probable that these beds may become the basis of an important industry.

About nine-tenths of the Lake Superior copper now comes from the mines of Houghton County, from beds of amygdaloid and conglomerate. These mines are worked on a large scale, and the stamp mills, run day and night, have a correspondingly large capacity, treating from three hundred to four hundred and fifty tons per day. The following table gives some details with regard to the principal mills of Houghton County, and will serve to illustrate the large scale upon which these mills are operated, and the importance of the copper-dressing industry:

Name of mill.	Character of rock.	Tons per 24 hours.	Yield of rock. Per cent.	No. of stamps.	No. of jigs.	Slime washers.
Calumet.	Conglomerate	800 to 900	4.5 to 4.75	4 Ball stamps.	96 ?	4 Ellenbecker tables.
Hecla.	Conglomerate			4 Ball stamps.	96 ?	6 Ellenbecker tables.
Osceola.	Conglomerate and Amygdaloid.	288	1.76*	2 Ball stamps.	84	5 Evans tables.
Quincy.	Amygdaloid.	280	1.90†	72 stamps 1000 lbs.	96	4 Evans tables.
Franklin.	Amygdaloid.	350	1.20‡	4 Ball stamps.	96	4 Evans tables. 2 Davey tables.
Atlantic.	Ash-bed.	400	0.96§	4 Ball stamps.	140	8 Evans tables.
Allouez.	Conglomerate.	210	1.40	2 Ball stamps.	56 ?	2 Evans tables.

* Yield of rock in 1878 (mostly conglomerate).
† Average five years (including a small amount of mass copper, about 4 per cent. of product).
‡ Average four years.
§ Average five years.

Character of the "Copper Rock."—Considered from an ore-dressing standpoint the amygdaloid and the conglomerate copper rocks present somewhat different characteristics. The most important point of difference is found in the size and character of the particles of copper disseminated through the rock. In the conglomerate the greater part of the copper exists in an extremely fine state of division, and the proportion of fine flat scales and of leaf copper is very large. In the amygdaloid rock, on the contrary, the copper is comparatively coarse. The copper from the so-called ash-bed of

the Atlantic mine, in this respect, stands between that from the amygdaloid and that from the conglomerate. The following table, showing the percentage of ingot copper obtained as "mineral,"* of different degrees of fineness, will illustrate this:

Grades of mineral.	Quincy amygdaloid.	Atlantic ash-bed.	Osceola conglomerate.	Allouez conglomerate.
Copper in large pieces, "heads" and "masses."	46.7	1.3	0.5	0.0
Coarse copper, as "No. 1" or "No. 2" mineral.	43.2	56.9	33.1	35.8
Fine copper, from "No. 2" and "No. 3," etc.	8.1	35.3	46.3	51.4
Finest copper, from "No. 4," "No. 5," etc.	2.0	6.5	20.1	12.8
	100.0	100.0	100.0	100.0

While the grades of copper obtained at the different mills are not strictly comparable as to size of grain, the mineral being graded according to its percentage of copper, and not according to its coarseness or fineness, still the above figures will represent pretty fairly the character of the copper in the conglomerate and the amygdaloid. We may assume, for purpose of comparison, that about 65 per cent. of the copper in the conglomerate rock will be in the form of particles less than a millimeter (or $\frac{1}{25}$th inch) in size. In the rock from the Atlantic mine we have but 42 per cent. of equally fine copper, and in the Quincy rock only 10 per cent.

Coarse copper is very easily saved; indeed, it can only be lost by most unskilful working or gross carelessness. The losses in copper dressing are almost wholly due to fine copper, which is either carried off as float copper, or in the form of fine particles included in the grains of coarse and fine sand. The great difference in the proportions of coarse and fine copper in the product obtained from the amygdaloid and in that from the conglomerate copper rock, would lead us to expect to find similar differences in the losses in the dressing of these rocks. We have only to compare the assays of the amygdaloid tailings of the Quincy mill with those of the conglomerate tailings from the Osceola, the Allouez, or the Calumet and Hecla mills, to see that this is the case, and that the losses are proportional to the fineness of the copper.

Oxide of Iron, Metallic Iron.—Associated with copper, in both the

* The so-called mineral is the product of the jigs, tables, and other ore-dressing apparatus, and is a mechanical mixture of copper, gangue, metallic iron, and iron oxide.

amygdaloid and conglomerate, is a certain percentage of specular iron (and of magnetite ?). To this is added some metallic iron, from the abrasion of the stamp shoes and mortar linings in crushing the rock, and from fragments of drills and other mine tools which find their way into the stamp mortars. The high specific gravity of these impurities cause them to separate with the copper, in the operations of dressing, from the comparatively light gangue; and the small difference in gravity between them and the copper makes it difficult or impossible to purify the resulting mineral by mechanical means. The greater part of this specular iron is already in, or is reduced by the stamps to a fine state of division, so that this impurity is chiefly found in the finer grades of copper, which for this reason are much less pure than the coarser products of dressing.

Oxide of Copper.—It is the commonly received opinion among many millmen of the Lake Superior region that a large part of the loss in the treatment of the fine slimes is due to the presence of suboxide of copper. Several tests were therefore undertaken with a view to determine the presence or absence of oxide of copper in the stamp rock and tailings.

Three methods have been proposed for the determination of suboxide of copper in the presence of metallic copper. 1st. By ignition in a current of hydrogen, reducing the suboxide of copper, and weighing the water produced. In the presence of oxide of iron this method would evidently not be applicable. A second method which has been employed, is based on the solubility of the suboxide of copper in dilute hydrochloric acid, which does not attack metallic copper. In the presence of iron, however, this method also is likely to give inaccurate results, as the perchloride of iron formed would probably dissolve more or less metallic copper. A third method,* based on the solubility of metallic copper in a neutral solution of nitrate of silver, seemed to be the only one available, and was therefore adopted.

First experiment: A sample of sand was taken from the launders of stamps No. 2 and No. 4 at the Atlantic mill. To avoid oxidation the sample was carried, covered with water, to the laboratory, weighed wet, about 1000 grains being taken for the test (the water being determined in another similar sample), and pulverized under

* Suggested by Mr. M. B. Patch, chemist to the Detroit and Lake Superior Smelting Works. These and other tests were made in the laboratory of the smelting works, and I must take this opportunity to acknowledge most heartily the many favors received from Mr. Patch and the other officers of the works.

water until fine enough to pass through a 100-mesh sieve. The resulting slimes were then treated for eight days with a neutral solution of nitrate of silver—(subsequent experiments proved that this long treatment was unnecessary, the solution of the metallic copper being generally completed in a few minutes),—which with the dissolved copper was then carefully decanted, and the sands thoroughly washed. Finally the washed sands were treated with nitric acid, and the copper in the solution determined by the colorimetric method. The result was as follows:

Copper insoluble in nitrate of silver,	0.52	per cent.
Copper, soluble in nitrate of silver, not determined, but probably about,	0.96	"
Total copper,	1.48	"

The amount of copper insoluble in nitrate of silver, presumably oxide of copper, silicate of copper, or some copper salt, agrees closely, as will presently appear, with the amount of copper in the tailings of the Atlantic mill. This coincidence at least suggests, though it does not prove, that the loss may be due wholly to oxide of copper.

The question at once arises: Does this oxide of copper exist as such in the copper rock, or is it produced after the rock is mined by the action of the moist air of the mines, aided by the gases of the powder used in blasting? The finely divided and chemically pure copper of the copper rock would certainly be liable to oxidation. Specimens of native copper, even though carefully protected in cabinets, soon become tarnished, and the finer grades of copper from the slime tables are said to turn quite black after having been barrelled a few weeks.

To test this question specimens of clean, freshly mined rock were taken from the lowest level of the Atlantic mine, 700 feet below the surface, the outside of each fragment removed by careful chipping, and the remainder crushed fine enough to pass through a 100-mesh sieve. Treated with a neutral solution of nitrate of silver, as above, the following results were obtained:

Metallic copper in coarse grains (sifted out),	1.36	per cent.
Fine copper, dissolved in nitrate of silver,	0.23	"
Copper insoluble in nitrate of silver,	0.09	"
Total,	1.68	"

It is possible that the small amount of copper not dissolved in nitrate of silver, may have been in the form of microscopic grains of metal included in and protected by particles of gangue. Or it may

have been oxide of copper formed during the pulverizing of the sample in the laboratory. So far as it can be proved by a single assay it would seem, therefore, that the freshly mined copper rock contains little or no oxide of copper; while after the ore has been mined and exposed, as is the case in the Atlantic mine, for several months in the stopes to the oxidizing action of the moist atmosphere of the mine, charged with powder gases, carbonic acid, etc., the proportion of oxide of copper becomes quite large, fully a third of the copper in the sample from the mill being insoluble in nitrate of silver.

The following table gives the result of these and other tests:

SAMPLES TESTED.	Copper dissolved by $AgNO_3$	Copper insoluble in $AgNO_3$	Total copper in sample.
Specimen freshly mined rock, Atlantic mine...	1.59	0.09	1.68
Sands direct from stamps, Atlantic mill..........	0.96 ?	0.52	1.48
Tailings from Evans table, Osceola mill...........	0.092	0.433	0.525
Heads from Evans table, Osceola mill...............	27.027	5.600	32.627
Waste sands from lake, Osceola mill...............	0.12	0.37	0.49

This method of determining suboxide of copper in the presence of metallic copper has, I find, been made the subject of an elaborate investigation by W. Hampe,* who finds that when chemically pure suboxide of copper is treated with a neutral solution of nitrate of silver, under the proper precautions, the following reaction takes place:

$$3\,Cu_2O+6AgNO_3+xH_2O=$$
$$2\,Cu_2H_3NO_6+2\,CuN_2O_6+6Ag+(x-3)\,H_2O.$$

That is, two-thirds of the suboxide of copper is converted into insoluble subnitrate of copper, while the other third is dissolved as soluble nitrate with the precipitation of silver.

C. Rammelsberg,† as the result of a single test, found, on treating a sample of pure suboxide of copper with neutral nitrate of silver, that 28.8 per cent. of the copper was dissolved, instead of 33.3 per cent.

Hampe, however, in a reply‡ to this criticism, points out some

* Zeit. f. Berg, Hütten, u. Salinenwesen, 1873, XXI, s. 218; Zeit. f. Berg, Hütten, u. Salinenwesen, 1874, XXII, s. 93; Zeit. f. Anal. Chemie, XIII, s. 176 and s. 352.

† Berichte d. Deutschen Chem. Gesellschaft, 12 Nov., 1877, No. 16, s. 1780.

‡ Zeit. f. Anal. Chemie XVII, 1878, s. 127.

discrepancies in Rammelsberg's results, and reaffirms his previous statement, which was based on a large number of analyses.

Whether 28.8 per cent., or 33.3 per cent. of the suboxide is dissolved, will, however, make but little difference in the result so far as the small quantities of oxide under discussion are concerned. An examination of the results of the tests of the tailings from the Evans table, and of the waste sands from the lake shore at the Osceola mill, will show that either allowance is more than sufficient to account for the copper dissolved by the nitrate of silver. Assuming this insoluble copper to be in the form of oxide, and correcting the results of the assays, we have—

SAMPLES TESTED.	Per cent. metallic copper.	Per cent. copper as oxide.	Total per cent. of copper.
Freshly mined rock, Atlantic mine..........	1.54	0.14	1.68
Sands direct from stamps, Atlantic mill..........	0.70 ?	0.78	1.48?
Tailings from Evans table, Osceola mill............	nil.	0.53	0.53
Heads from Evans table, Osceola mill...............	24.23	8.40	32.63
Waste sands from lake, Osceola mill................	nil.	0.49	0.49

From this table we see that the amount of oxide of copper in the freshly mined rock is quite small, and that by exposure to oxidizing influences for several months in the mine, the amount of oxide is increased till half the copper in the rock is converted into oxide. The assay of the heads from the Evans table shows that a large part of this oxide is saved in the slime treatment, and no doubt assays of the fine copper from the finishing jigs would also show oxide of copper saved by these jigs. The specific gravity of oxide of copper is 6.0, high enough to make possible its separation from the gangue by dressing, provided that its friability does not cause it to be reduced in the stamps to a slime too fine to be saved.

The assay of the waste sands from the edge of the lake at the Osceola mill shows that all the included copper in these sands has been converted to oxide of copper, and from the small percentage found it seems not improbable that a part of the copper has been converted into a soluble salt and leached out. A single assay, however, cannot be taken as a conclusive proof of this; and, indeed, the conclusions as to the amounts of oxide of copper present, and the effect of oxidizing influences require further investigation.

It is possible that a part of the copper found to be insoluble in

nitrate of silver may have been metallic copper of such microscopic fineness as to be included within and protected by the very small particles of gangue passing through a 100-mesh sieve. Copper thus protected would not be attacked by nitrate of silver; but when treated with nitric acid, by the solution of the melaphyr gangue, the copper would be liberated and dissolved, thus behaving as oxide of copper in all respects. Polished specimens of copper rock frequently show copper in an almost microscopic state of division, and finely disseminated through the rock.

Some of the tests made with a view to determine the amount of included copper in sands of different degrees of fineness would seem to support this hypothesis, and it is not unlikely that much of the loss now attributed to oxide of copper may be due to included metal. That the insolubility in nitrate of silver of the copper is wholly due to this cause seems doubtful; first, on account of the great difference in this respect between the freshly mined rock and the same rock after several months' exposure to oxidizing influences; and secondly, the amount of this insoluble copper is in most cases too great to be accounted for as included copper. If the copper insoluble in nitrate of silver were included copper, we should find about the same proportion of free and included, or of soluble and insoluble copper, in freshly mined rock, in rock several months mined, and in sands exposed on the lake shore for a year or more. The table on the previous page shows, however, a diminution of free copper, both in percentage and in amount, which can only be accounted for on the supposition that the metallic copper is oxidized by exposure.

In the second place, metallic copper included mechanically in grains of gangue will be only partially protected from the action of a solvent; that is, the grains on the surface of the particle of gangue, and thus only partly covered, would form a large proportion of the included copper. So, if we suppose the copper to be wholly in the metallic form, the mechanical assay, by vanning, should show more included copper than the chemical test, as in the latter case a part of the included copper would be dissolved. To test this, comparative assays by the two methods should be made on the same specimen, with other assays to test the effects of finer comminution on the amount of copper dissolved by nitrate of silver.

Whether we consider the copper insoluble in nitrate of silver as oxide of copper or as microscopic particles included in grains of gangue, the fact remains that both in the copper rock from the conglomerate mines, and in that from the so-called ash-bed of the Atlantic

mine there is a large percentage of copper, the saving of which by mechanical means is difficult or impossible. What steps should be taken to diminish the loss from this cause can only be determined by further investigation.

If the loss be due to oxidation of copper after mining, it is probable that it may be lessened by taking precautions to diminish or prevent the oxidation of the copper by milling the rock as soon as possible after it is mined, and not allowing it to remain so long in the stopes. In extreme cases a portion of the rock may remain as long as a year on the stulls, before being sent to the mill. By adopting underhand stoping, instead of the method of overhand or "back" stoping used, the rock might be sent to the surface as soon as mined. Again, the copper exposed to oxidizing action will be much greater in the fine ore than in the large pieces. It may therefore prove advisable to encourage the miners to use less powder, and to bring down the ore in large pieces, and, possibly, even necessary to establish a system of fines and penalties to prevent the production of fine ore and dirt, as is the custom in the coal mines. Finally, the nature of the explosive used may prove to have influence on the amount of oxide of copper produced. Hercules powder, so largely used in many of the copper mines, contains a very large proportion of nitrate of soda, which probably serves no useful purpose in increasing the strength of the powder, the remainder of the dope being incombustible carbonate of magnesia. This large excess of nitre not being utilized to burn sulphur, carbon, or other oxygen-consuming ingredients as in other powders, may, and probably does, oxidize a certain amount of copper. The loss from this cause may more than counterbalance the advantage derived from the use of this explosive.

If, on the other hand, the copper insoluble in nitrate of silver be proved to be finely disseminated copper, the remedy will be quite different. It will consist in fine crushing, increasing the production of slime, necessitating more slime tables in the mills, and improved slime treatment.

The determination of this question becomes thus of vital importance. The saving of half the copper now lost would add largely to the profits of the mills, and should the price of copper fall again to the low figures which have prevailed in the last year or so, the solution of this problem may mean, in many cases, success instead of failure, and profit where otherwise there would be a ruinous loss.

Copper Dressing, Methods Used, etc.—The method of dressing employed in the Lake Superior mills may be called the English or Cornish system. It has been gradually developed in the copper region from the simple methods of hand dressing, formerly used in the tin mines of Cornwall, and brought to this country by the Cornish tin dressers. The chief characteristics of the method are, first, the absence of any preliminary sizing of the sands by screening; and second, the jigging of the mineral through a bed of coarse copper on the sieve of the jig, and through the meshes of the jig-sieve into the hutch below. The apparatus and methods employed have already been described in some detail by Mr. Charles M. Rolker,* in a paper read before the Institute; later by Professor Egleston,† in the *Metallurgical Review*, and also by Commissioner E. F. Althans,‡ in his valuable report on Mechanical Dressing, published in the Centennial reports. I shall therefore refer to these papers for details, and content myself with a brief résumé of the process.

The copper rock is stamped wet, with a large volume of water, through $\frac{3}{16}$ inch screens. The stream of sand and slime is roughly separated into coarse, medium, and fine sands in the hydraulic separator, the slime passing through and out at the end of the separator to the slime-boxes. The coarse, medium, and fine sands from the three or four spigots of the hydraulic separator go to the jigs. The stream of sand from each spigot passes in succession over the sieves of two jigs, by which all the free copper is separated, and then, without further treatment, into the waste launders as "rough sands." The finer and heavier portions of the sands, with the fine copper, sink through the bed of coarse copper on the jig sieves, and passing through the sieves are collected in the hutch or jig-box below. This hutchwork, passing out through spigots in the bottom of the hutch, is carried by launders to the "finishers," a series of two or three jigs, over the sieves of which these hutchwork sands pass in succession. These hutchwork sands are, of course, quite rich, and even after passing over three sieves, they sometimes contain considerable free copper, especially in the finer grades. In the Franklin and in other mills these sands are allowed to flow through large catch-boxes, in which the heavier portions settle, which boxes are emptied from time to time and the contents treated on a hand-

* Transactions, vol. v, p. 584.

† Metallurgical Review, vol. ii, pp. 227, 285, 389.

‡ International Exhibition, 1876, Reports and Awards, Group I, pp. 257, 289, 309, and 334.

buddle, or on the sieves of an extra set of jigs, provided these are not otherwise employed. At the Atlantic mill these sands go through launders to revolving tables (Evans tables), pitched somewhat steeper than the slime-tables.

The greater part of the copper is brought by this simple treatment into a concentrated form, the so-called mineral, which is sent to the smelting works. There is, however, a certain quantity of intermediate product, coming from different sources, which is not rich enough to be smelted, and yet too rich to be thrown away. This is enriched by treatment in keeves, hand-buddles, and other auxiliary washing apparatus; giving rise to other intermediate products, fortunately in small quantity, which resist all attempts at concentration. This mineral consists mainly of very fine copper, partly in flat scales—*i. e.*, very light "float copper"—associated with sands rich in included copper, and grains of metallic iron, iron sand, and other heavy minerals. When vanned on a shovel with great care, most of the free copper in this mineral can be obtained as a head, but if in vanning water be allowed to sweep gently over this head, the float copper will be swept over the sands, and we shall have on the shovel a copper head and a copper tail, with sand between. At the Osceola mill, after many attempts to concentrate this product,* it is now sent to the smelting works with other similar products, as "low grade mineral" or "X mineral," averaging 10 to 20 per cent. of copper.

The tailings from the mills may be divided as follows:

1. *Rough sands,* coming from the head or roughing jigs, having passed over the sieves of two jigs.

2. *Fine sands,* coming from the "finishers" or jigs treating the hutchwork of the roughing jigs.

3. *Slimes,* tailings from the Evans table or other slime washers.

4. *Fine slimes,* overflow from the slime-boxes which supply the slime-tables. This is slime so fine that it does not settle in any of the boxes, but passes through the mill without treatment.

These tailings may contain copper in one or all of the following forms:

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* Treated on a Frue vanner the tails contained 9 per cent. of copper, or nearly as much as the heads. Tossed in a keeve the top skimmings contained 5.2 per cent., the second skimmings 9 per cent., and the bottom 13 per cent. In 1877 a few tons only of this mineral were sent to the smelting works, averaging 19 to 20 per cent. of copper. In 1878, special arrangements having been made with the smelting works, the Osceola mill produced over 300 tons "X mineral," averaging 13¼ per cent. of copper, yielding nearly 40 tons of ingot.

1. *Float copper*, in the form of fine grains and flat scales, free from gangue.

2. *Included copper*, in the form of fine particles, included in grains of gangue.

3. *Oxide of copper*, which may be either included in grains of gangue or free.

The rough sands should contain little or no float copper, provided that the hydraulic separator is working properly, as copper which is heavy enough to fall through the openings in the bottom of the separator, against the rising current of clear water, will also find their way through the sieve of the jig into the hutch below.

The fine sands from the finishers usually carry off more or less float copper. The sand tables in the Atlantic mill, treating 85 to 90 tons of these sands per day, save about 40 tons of mineral, or nearly 15 tons of ingot per year, saving thus about 0.05 per cent. copper which would otherwise be lost.

The slimes from the tables and from the settling-boxes also carry off a trace of float copper, but in an extremely fine state of division.

The rough sands, fine sands, and even the slime contain included particles of copper, the percentage varying with different copper rocks, and increasing with the fineness of the copper and the coarseness of the sands. This included copper can only be separated and saved by fine crushing or grinding of these sands or slimes, and treatment of the crushed material.

Sampling of Sands, Assays, etc.—It is by no means an easy matter to obtain an average sample of tailings for assay. Thirty-five, forty, fifty, and even sixty tons of water are used in the mill for every ton of rock treated. The waste launders are filled with a swift, turbulent stream, a part of the sand being held in suspension moving with the water, while the heavier and coarser sands move more slowly at the bottom of the trough. An average sample must be taken from the whole current, including the proper proportion of the water as well as of the sand, and this sample must be allowed to settle quietly for twelve hours or more before the water is drawn off. Otherwise the float copper and the finest slimes will be carried off, and the assay will show only the included copper contained in the coarse sands. At the Quincy mill a sample is taken daily from the elevating wheel, which raises the tailings to the launder leading to the tail-house. Once a

day a box is placed in a position to catch the drippings from the wheel, just beyond where the buckets empty themselves. The box is moved from time to time to take the drip at different points. When the box is full the water is rudely decanted and a portion of the sands put in the sample-box, from which box an average sample is taken once a month for assay. The float copper and the fine slimes are thus lost, and the assay probably does not show all the copper lost in the tailings. The copper in the Quincy rock is, however, extremely coarse, as we have already seen, so that it is doubtful whether much, if any, float copper passes the jigs. The same method of sampling is said to be used at the Calumet and Hecla mills, where, on account of the extreme fineness of the copper, it is evidently not well adapted to the purpose. Assays of such samples can only serve to mislead.

In order to obtain satisfactory samples, a mechanical sampling apparatus should be devised, which will divert a section of the stream in any desired launder into a settling-box. This apparatus should either work continuously or be arranged to take samples at short intervals. By a series of automatic arrangements of this character the sample could be reduced to manageable proportions, even though the apparatus worked continuously.

In default of such samplers, the specimens tested were taken in such way as seemed best adapted to secure average results. In testing the working of tables and similar apparatus, samples were taken with a vanning shovel at different places, care being taken not to allow the shovel to overflow, and the contents emptied into a clean pail and allowed to settle. In taking samples from the launders at the Atlantic mill, a short length of 2-inch rubber hose was used to divert a part of the stream into a clean barrel, the hose being moved about in the stream, and the barrel allowed to stand till the fine sands and slimes had completely settled. In taking samples of the waste sands from the lake, at this and other mills, a small pit was dug, and a slice taken from top to bottom on one side, thus securing a sample representing the growth of the bank; the deeper portion being that deposited at some distance from the shore while the bank was forming under water, and the upper layer that more recently deposited above water and near the end of the launder.

The attempt to take average samples from a launder with a hose was not very successful. Careful tests of the average fineness of the samples thus taken, by sifting through sieves of different mesh,

proved that more than the average proportion of fine sand and slime was obtained.

The amount of copper was determined by the colorimetric method, which in skilful hands gives very accurate results, and which is especially well adapted to the determination of the small quantities of copper, usually less than one per cent., found in these sands. Mr. Patch, in whose laboratory the assays were made, has confirmed the accuracy of the method by numerous tests, and my assays were in most cases checked by him, with closely agreeing results.

Included and Float Copper.—The following tests were made on a sample of conglomerate tailings from the Calumet mill, to determine the amounts of included and float copper. The sample was taken from the edge of the waste sands in the lake near the end of the launder, observing the precautions already described, in order to obtain an average sample. The sands were sifted under water through a 20-mesh sieve, and the siftings through a 50-mesh sieve. The portion passing through the 50-mesh sieve was washed several times by decantation to separate the slime too fine to settle within three minutes. The wash waters were mixed and allowed to stand three minutes, and then decanted and allowed to stand over night. The percentage of material remaining on and passing through the different sieves was as follows:

Over 20 mesh, $\frac{1}{25}$ to $\frac{3}{16}$ inch (1 mm. to 4.6 mm.),	28	per cent.
Over 50 mesh, $\frac{1}{64}$ to $\frac{1}{25}$ inch (0.4 mm. to 1.0 mm.),	20	"
Through 50 mesh, less than $\frac{1}{64}$ inch (less than 0.4 mm.),	50	"
Slimes, not settling in 3 minutes,	2	"
	100	"

The different samples being tested for copper yielded as follows:

Coarse sands, 1.0 mm. to 4.6 mm.,	1.48 / 1.38	per cent. / "
Fine sands, 0.4 mm. to 1.0 mm.,	1.15	"
Slimes, less than 0.4 mm.,	2.70	"
Finest slimes, not settling in 3 minutes,	1.25	"
Average,	2.04	"

A portion of each sample was carefully vanned by an expert. The coarse and the fine sands yielded no free copper, proving that all the float copper is less than 0.4 mm. The fine sands or slimes passing through the 50-mesh sieve yielded to the first vanning 1.85 per cent. of free copper. These heads were carefully vanned, to separate the heavy from the light copper, and thus determine what proportion of

the free copper can be saved by mechanical means, and to what extent the copper can be concentrated. The tails of the first vanning were revanned, producing a small quantity of low grade mineral. 2600 grains of sand were thus treated. Reduced to product per 100 grains, the results may be summed up as follows:

PRODUCTS OF VANNING.	Containing copper.	Grains copper.	
1st vanning, 1.94 grains concentrated heads	43.9 per cent.	0.85	
7.30 grains, middle heads	13.7 "	1.00	
2d vanning, 2.30 grains heads	7.0 "	0.16	
Total free copper in fine sands and slimes			2.01
84.00 grains tailings	0.71 per cent.	0.60	
4.46 grains loss in vanning		0.09	
Total included copper in fine sands and slimes			0.69
100.00 grains sand	2.70 per cent.		2.70

It will be seen from this table that these fine sands or slimes contain about 0.60 to 0.69 (?) per cent. of included copper, which cannot be separated by vanning, and over 2 per cent. of free copper. Of the free copper fully half cannot be concentrated beyond 13 or 14 per cent. without loss; this probably consists largely of sand rich in included copper, associated with heavy minerals and some very light float copper. It would be interesting to test the effect of grinding this low grade mineral, as it is probable that by this means it might be rendered capable of concentration.

The very fine slime, separated by washing, and containing 1.25 per cent. of copper, was not vanned, but it is probable that it contains nearly one per cent. of free copper, as material of equal fineness obtained in other experiments, and known to contain little or no free copper, yielded but 0.30 included metal.

From the above experiments we find that the Calumet waste sands contain:

Free copper, in grains. 0.4 mm. and less, . . .	0.434 per cent.	
Fine copper, to be saved with difficulty, . .	0 610 "	
Total free copper, ...		1.044
Included copper, disseminated through grains of coarse and fine sand, etc., ...		.998
Total copper, ...		2.042.

The percentage of included copper is greatest in the coarse sands, and least in the fine slimes, as is shown in the following table of assays of vanned sands free from float copper:

Included Copper in Calumet Sands and Slimes.

Coarse sands (1.0 to 4 6 mm.),	1.48 per cent. 1.38 "
Fine sands (0.4 to 1.0 mm.),	1.15 "
Slimes (0.2 to 0.4 mm.),	1.03 "
Slimes, less than 0.4 mm.,	0.71 "
Slimes, less than 0.2 mm.,	0.69 " 0 66 "
Slimes (not settling in 5 minutes),	0.40 " 0.30 "

The finer the sands, therefore, the less included copper they contain. By crushing any of these sands or slimes still finer a certain amount of copper will be liberated, which can be saved by further washing. The percentage of the included copper saved in this way will increase with the fineness of the crushing; provided, of course, that the crushing be not carried so far as to reduce the copper to the form of float, difficult or impossible to save.

To test this matter by direct experiment, five hundred grains of coarse sands were crushed in an iron mortar fine enough to pass through a 100-mesh sieve. The finest slimes were then removed from the crushed material by washing, the remainder vanned, and the amount and percentage of copper in the different products determined by assay. The results were as follow:

			Grains copper.
18 5	grains	heads (very fine copper), @ 25.10 per cent.,	4.65
347.0	"	tails (0 2 mm and less), @ 0.69 "	2.40
134.5	"	slimes (not settling in five minutes), @ 0.30 per cent.,	0.35
500.0	"	sands, @ 1.48 per cent.,	7.40

From this it appears that by crushing the coarse sands to the above fineness, about sixty-three per cent. of the copper is liberated, and may be saved. The percentage of copper in the tailings, including that in the fine slimes, may thus be reduced from 1.48 to 0.57. By crushing still finer more copper will be liberated, but at the same time this copper will be exceedingly fine and difficult to save. The economical limit of fine crushing can, of course, only be determined by practical experiment on a large scale. It is evident, however, that nothing less than the production of a fine slime, and the introduction of an efficient slime treatment, will

save any notable amount of the copper now carried off in the Calumet sands.

A similar experiment was made on still finer sands, namely, those passing through a 50-mesh sieve (from 0.2 to 0.4 mm. diameter), all grains passing through a 100-mesh sieve (less than 0.2 mm. diameter) having been previously separated. In this case, by crushing all fine enough to go through a 100-mesh sieve, about forty-seven per cent. of the included copper was freed and separated by vanning, and the included copper in the tailings reduced from 1.03 per cent. to 0.66 per cent.

It seems probable, therefore, that even the very finest slimes, could they be crushed still finer, would yield free copper; a strong argument in favor of the hypothesis that the so-called oxide is in great part finely disseminated metallic copper, locked up in minute grains of gangue.

The following assays of the tailings of other mills treating conglomerate rock are of interest, although, as no attempt has heretofore been made to separate the free and float copper from the included copper, they are of little value for comparison with the results obtained in the tests of the Calumet sands. Indeed, from the manner in which the samples of sand for assay have generally been secured, it is doubtful whether these results fairly represent the losses at the different mills, the samples containing probably little or no float copper.

ASSAYS OF CONGLOMERATE TAILINGS.

Name of mill and assayer.	Coarse sands.	Fine sands.	Slimes (from tables).	Slimes (overflow).	Misc. sands.	Float copper.
Allouez mill, 1875. C. M. Rolker.	1.32 1.21 0.87	1.03 0.98 0.84	0.90 0.82 0.78		tail-house 1 36	un-determined.
Allouez mill, 1876. M. B. Patch.	1.10 0.90	1.30	0.45			un-determined.
Allouez mill, 1879. E. E. Olcott. C. D. Bolton.	0.40 0.62	1.09 0.85				un-determined.
Osceola mill, 1877. M. B. Patch.	1.16 0.90	1.20 0.90	0.60		tail-house 1.11	un-determined.
Osceola mill, 1879. H. S. Munroe.			0 34† 0.525†	0.40†	lake 0.49*	traces only.
Calumet mill, 1879. H. S. Munroe.	1.48* 1.38*	1.15*	0.85† 0.88†		lake 2.042†	1.044‡

* Included copper only. † Included and float copper. ‡ Total free copper.

The following results were obtained from the tailings of the Atlantic mill, treating very poor rock from the so-called ash-bed, the copper being intermediate in fineness between that of the conglomerate beds and the coarse copper of the amygdaloid beds:

ASSAYS OF TAILINGS FROM ATLANTIC MILL, H. S. MUNROE, 1879.

TAILINGS.	Included copper.	Float copper.
Rough sands, from roughing jigs........	0.425	trace.
Fine sands, from finishing jigs...........	0.470*	0.050†
Slimes from Evans table......................	0.460	trace.
Slimes, overflow from settling-boxes...	0.200	none.
Coarse sands from lake.........................	0.540	not determined.

* After retreatment on Evans tables. † Copper saved by tables.

At the Quincy mill, treating amygdaloid rock yielding coarse copper, the losses are quite small. At this mill assays are made every month on an average sample made up of samples taken daily from the elevating wheel which raises the mill tailings to the launder leading to the tail-house. The sample is sifted through sieves of different mesh, and the portions remaining on each sieve assayed separately. The following table gives the results of the assays for six months. The method employed for taking the samples is defective, as has been already pointed out; but, owing to the coarseness of the copper in the amygdaloid rock treated, it is probable that there is but little float copper in the tailings. The results of the assays, therefore, probably would be changed but little by a more careful taking of the samples:

ASSAYS OF TAILINGS OF QUINCY MILL (AMYGDALOID), H. C. SOUTHWORTH, 1878–79.

Date.		Side of mill.	6-mesh sieve.	8-mesh sieve.	12-mesh sieve.	16-mesh sieve.	20-mesh sieve.	Fine thro' 20.
1878	November.	N.	0.35	0.28	0.33	0.22	0.15	0.17
		S.	0.34	0 25	0.30	0.31	0.27	0.19
1878	December.	N.	0.16	0.26	0.29	0.20	0.26	0.19
		S.	0.46	0.21	0 29	0.23	0.25	0.23
1879	January.	N.	0 36	0.13	0.26	0.16	0.18	0.17
		S.	0.23	0.16	0.21	0.23	0.21	0.19
1879	February.	N.	0.10	0.16	0.31	0.18	0.18	0.18
		S.	0.34	0.44	0.36	0.13	0.21	0.17
1879	March.	N.	0.23	0.20	0.15	0.18	0.18	0.15
		S.	0.15	0.15	0.26	0.25	0.13	0.16
1879	April.	N.	0.13	0.15	0.32	0.27	0.20	0.20
		S.	0.25	0.23	0.18	0.21	0.23	0.22
	Average,		0.258	0.220	0.272	0.214	0.204	0.185

The following tables, calculated from the results of the above assays, from the returns of the smelting works, and from other data, give approximately the amounts and percentage of metallic copper obtained as mineral, and lost in the tailings at the different mills, in the treatment of 1000 tons of copper rock:

CALUMET MILL (CONGLOMERATE).

TAILINGS.	Tons of rock.	Yield ingot copper (pounds).	Yield per ct.	Per ct. of copper in tailings.		Pounds of copper in tailings.		Loss per ct.
				Included.	Free.	Included.	Free.	
Sands, coarse and fine.	650?	86,800	9.64	{ 1.43 { 1.10	2.54	16,048	18,780	28.0
Slimes	350?	4,200	0.60	0.56	0.30	3,920	2,100	60.0
Totals	1000	90,000	4.50	0 998	1.044	19,968	20,880	31.0

OSCEOLA MILL (CONGLOMERATE).

TAILINGS.	Tons of rock.	Yield ingot copper (pounds.)	Yield per cent.	Per cent. of copper in tailings.	Lbs. of copper in tailings.	Loss per ct.
Rough sands. } Hutchwork sands. }	546	32,800	3.00	1.05	11,465	25.9
Slimes, from tables	294	2.400	0.41	0 50	2,940	53.2
Slimes, overflow	160	nil.	nil.	0.40	1,280	100.00
Totals	1000	35,200	1.76	0.78	15,685	30.8

Of the copper lost in the tailings at the Osceola mill in the treatment of 1000 tons of rock, the float copper will probably be less

ATLANTIC MILL (ASH-BED).

TAILINGS.	Tons of rock.	Yield ingot copper (pounds).	Yield. per ct.	Per ct. of copper in tailings.		Pounds of copper in tailings.	Loss per ct.
				Included.	Float.		
Rough sands	350	} 17,950	1.632	0.425	Undetermined. Small traces only.	2975	} 21.3
Hutchwork sands	200	}		0.470		1880	}
Slimes, from Evans tables..	270	1,250*	0.231	0.460		2484	66.5
Slimes, overflow	180	nil.	nil.	0.200		320	
Totals	1000	19,200	0.960	0.383		7659	28.5

* Including about 200 pounds (of ingot) saved from hutchwork sands by Evans sand tables.

than 1500 pounds, or not more than a tenth of the total loss. At the Calumet mill the loss in float copper in the treatment of the

same amount of rock is probably over 20,000 pounds, or fully half the loss in the tailings is in the form of float copper. This difference, as will presently be shown, is chiefly due to the imperfect slime treatment at the Calumet mill.

Before leaving the subject it will be well to say a few words as to the value and importance of regular and systematic assays of the tailings, as a means of controlling and perfecting the working of the mills, and diminishing the losses in the tailings.

The perfection and improvement of the methods of dressing effected of late years, and the present high state of development of the dressing process in the different mills, as shown in the small losses in float copper, are due to the care and skill of the efficient copper washers in charge of the work, guided in their experiments by frequent and careful tests of the tailings on the vanning shovel. Too much cannot be said in favor of these mechanical assays, or of the value of the vanning shovel to those in charge of the mills. The most careful vanning, however, will not reveal the presence nor indicate the amount of included copper in the tailings.

In order still further to perfect the dressing process and diminish the losses, the copper washers must be guided by frequent and careful assays of their tailings. As the problem of how to save the float copper has been already solved, with the aid of the vanning shovel, so the saving of the oxide of copper and the included copper must now be sought in the light of daily, and even hourly assays of tailings, middle products, coarse and fine sands, slimes, rock from the mine, from the stamps, etc.

These assays will be of little value unless made on average samples. Some mechanical, automatic, and continuous sampling apparatus, that will take a perfect sample, must therefore be devised and used for the purpose. By the use of such sampling devices a perfect control of the operations of the mill may be secured, possible in no other way. The advantage of having samplers at work night and day, as a check upon careless or incompetent subordinates, need hardly be suggested.

At present the assays are of but little value, both because not made often enough to show variations in working when such occur, or the effect of changes when such are made, and also because the assays are not made upon average samples.

When those in charge of the mills become accustomed to the numerous advantages in the control of the work, and in the possibilities of improvement suggested by regular and systematic assays of

the tailings, it is not likely that they will wish to give them up. The time is probably not far distant when a well-appointed assay office shall be considered as essential to the business of the mill, as is the office when a record is kept of the dollars and cents expended.

Review of the Lake Superior Dressing Methods.

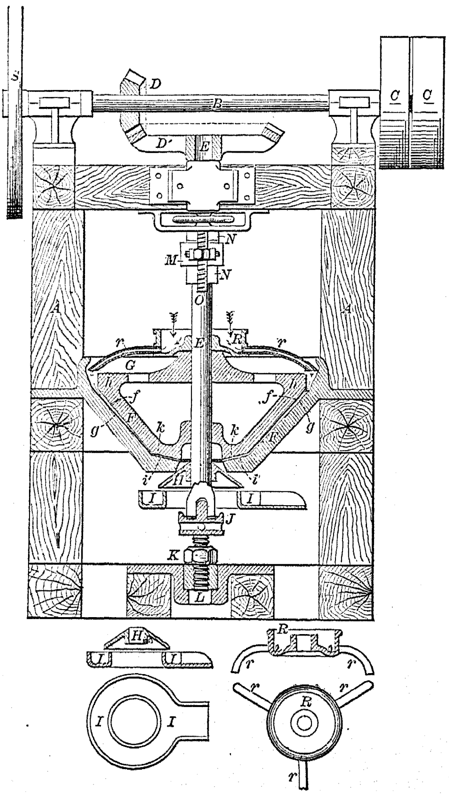
Crushing of the Rock.—We have already seen that the finer the rock is crushed the smaller will be the loss due to included copper. In the case of the Quincy mill, treating amygdaloid rock, it is evident from the assays that the crushing of the rock is fine enough, and that but little gain will result from finer crushing. It is even possible that coarser screens might be used with advantage. In the Atlantic mill, treating a finer-grained copper, the coarse and fine sands and the slimes contain practically the same percentage of copper. The very finest slimes only, from the overflow of the settling-tanks, show a reduction in the amount of included copper. This would seem to indicate that finer crushing will release but little fine copper unless the crushing be pushed so far as to reduce the rock to an impalpable slime. This is clearly impracticable. Fine crushing, short of the production of an impalpable slime, will simply increase the amount of fine copper difficult to save. Both finer and coarser stamp screens have been tried at this mill, and after long and careful experimenting, a screen with $\frac{3}{16}$-inch (4 mm.) openings was adopted, having been found to give the best results.

The case is quite different at the Osceola, the Allouez, and the Calumet and Hecla mills, treating conglomerate rock. From the experiments on the tailings of the Calumet mill, it is evident that the rock must be reduced to a slime in order that the copper may be separated. The heroic remedy at once suggests itself of replacing the present $\frac{3}{16}$-inch screen of the Ball stamps, by fine slotted screens, as used in the gold and silver mills of the West, with $\frac{1}{40}$ or $\frac{1}{50}$ inch openings. The jigs at present used would then be replaced by slime washers. There are several objections to this course. The stamps, with $\frac{3}{16}$-inch screens, now produce about 72 per cent. of slime (grains of 1 mm. and less), as is shown by the table below. To crush the other 28 per cent. to the same fineness it would hardly pay to double the time and cost of stamping, and reduce one-half the capacity of the mill by the use of $\frac{1}{25}$-inch screens instead of $\frac{3}{16}$-inch.

COARSENESS OF SANDS FROM STAMPS ($\frac{3}{16}$-INCH SCREENS).

	Quincy.	Atlantic.	Calumet.
Over 20 mesh (1.0 to 4.6 mm., or $\frac{1}{25}$ to $\frac{3}{16}$ inch grains).........	31.20	32.5	28.0
Over 50 mesh (0.4 to 1.0 mm., or $\frac{1}{64}$ to $\frac{1}{25}$ inch grains)	68.80	25.2	20.0
Through 50 mesh (less than 4 mm., $\frac{1}{64}$ inch)........................		42.3	52.0
	100.0	100.0	100.0

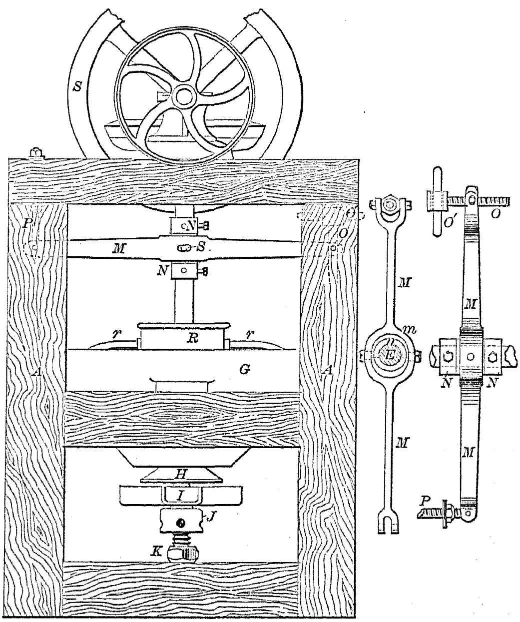
Again, as we have already seen, about 35 per cent. of the copper in the conglomerate rock is coarser than $\frac{1}{25}$th of an inch. Most of this would probably remain in the mortar of the stamp, accumulating at the rate of two and a half tons a day in each mortar, and



making it necessary to clean the mortar at short intervals. Or should this copper be ground up by the action of the stamps, the proportion of float would be largely increased.

It becomes at once evident that the best plan will be to recrush the sands, coarse and fine, after separating the coarse copper by jigging. To liberate any large proportion of copper, it will be necessary to use very fine screens in the stamps, say $\frac{1}{50}$th of an inch. By crushing to this fineness fully one-half of the copper will be liberated. The slimes from the Ball stamps and from these fine stamps can be treated together on Evans tables or other slime washers. The sands will contain about 1.3 per cent. of copper, and will yield, say 0.60 per cent., which should give a handsome profit.

At the Calumet and Hecla mills small stamps have been erected in the tail-houses, and are used to restamp a portion of the sands. At the time of my visit these stamps were treating about five or ten



tons of sands per day, previously concentrated by jigging. The screens used were quite coarse, having $\frac{1}{12}$-inch openings. It was proposed, however, to experiment still further with $\frac{1}{16}$th screens. It is evident, however, that still finer screens than these must eventually be employed.

At the Phœnix mill the Hodge grinder has been in use for some time, for the recrushing of the richer sands from the jigs, and is

apparently doing good work. The machine is said to have a capacity of twenty tons per day, and the wear of iron is claimed to be small.

The machine is shown in the accompanying drawings. It consists of a conical cast-iron rubber working in a funnel-shaped cast-iron pan. The bottom of rubber and pan are truncated and nearly flat, having a slight inclination toward the centre, to aid the discharge of the crushed sands. The sides of rubber and pan converge toward each other in the conical part of the apparatus. It is claimed in the patent that not only is the quartz or gangue rock thoroughly disintegrated, but that the leaf and scale copper is rolled into compact pellets, whereby the loss in float copper is lessened. I have specimens from the Phœnix mill illustrating this effect in a remarkable degree. Both fine and coarse copper is rolled into shot and cigar shaped pellets, quite dense, and easily saved. Whether this action extends to the microscopic grains, such as are found in the conglomerate rock, I am unable to say.

An experiment shown me by Mr. Evans, superintendent of the Atlantic mill, is of interest in this connection. Vanning, with great care, some of the tailings from one of the slime-tables, which was treating the fine sands from the finisher jigs, he separated as a head a small quantity of very light float copper. Then, by an easy motion, he permitted a thin film of water to sweep gently over the shovel. Instantly the head was swept as float copper over the sands on the shovel, showing how very light and thin were the scales of copper of which it was composed. Next, with a smooth piece of hard wood he rubbed vigorously the sands and the copper on the bottom of the shovel. On vanning again, the copper was easily brought to a head, and only a small portion, which had perhaps escaped the action of the improvised rubber, remained as float copper. The rest was quite heavy, and remained as a head clinging to the shovel, even under a stronger current of water than that which sufficed before to send it floating over the sands.

The machine at the Phœnix mill is not worked up to its full capacity, and has only been in use a few months. The question of the cost of crushing the sands, and how poor sands can be treated in this way, remains to be solved. About twenty tons of Calumet tailings were crushed in this machine as an experiment, and are said to have yielded $1\frac{1}{2}$ per cent. of mineral. Even did this mineral contain but 30 or 40 per cent. of copper, the result was a very favorable one. Whether the sands can be ground fine enough in one operation to release more than half the copper is of course doubtful.

The Hydraulic Separator.—The stream of sands and slimes coming from the stamps passes through the hydraulic separator, in which the slimes are separated from the sands, and the latter divided into three or four grades for jigging. The apparatus consists of a double V trough, a small trough within a larger one. The inner trough is from a foot to a foot and a half in width, and 10, 12, 14, or 15 feet long. The outer trough is 2, 2½, or 3 feet in width, and of the same length as the inner one. The inner trough is open at both ends, and through it the stream of slime from the stamps rushes in a swift, turbulent stream, at the rate of about thirteen cubic feet per minute.

The space between the inner and the outer troughs is divided into three or four compartments. These are kept full of clear water, the level of the water being maintained above that of the stream of slime. Through slits cut in the bottom of the inner trough the sands find their way, against a rising current of clear water, into these compartments, from which they are discharged by spigots to the jigs.

The following table shows the amounts of water and sand discharged per hour from the different spigots of the hydraulic separators, as determined by experiment at the different mills:

OSCEOLA MILL.

SPIGOTS OF SEPARATOR.	Cubic feet of water per hour.	Pounds of sand per hour.	Per cent. of sand from each spigot.	Per cent. of sand and slime.
1st spigot, coarsest sands	84.5	1300	43.5	54.5
2d spigot, medium sands	51.7	970	31.5	
3d spigot, medium sands	55.4	657	21.3	
4th spigot, fine sands	36.0	113	3.7	
Outflow at end of separator	776.0	2523		45.5
Total	1003.6	5563	100.0	100.00

ATLANTIC MILL.

SPIGOTS OF SEPARATOR.	Pounds of sand per hour.	Per cent from each spigot.
1st spigot	1716	66
2d spigot	546	21
3d spigot	234	9
4th spigot	104	4
Total	2600	100

FRANKLIN MILL.

SPIGOTS OF SEPARATOR.	Cubic feet of water per hour.	Pounds of sand per hour.	Per cent. of sand from each spigot	Per cent. of sand and slime.
1st spigot, coarsest sand............	88.7	1440	65.0	54.3
2d spigot, medium sand..............	92.5	471	22.0	
3d spigot, fine sands..................	54.7	291	13.0	
Outflow, slime..........................	not determined.	1904	.	45.7
Total..................		4106	100 0	100.0

About 54 per cent., therefore, of the product of the stamps is removed from the stream of slime by the hydraulic separator. The remaining 46 per cent. goes to the slime-boxes, and from these to the Evans tables. The sands from the hydraulic separator, going to the jigs, varies in size from 0.4 to over 4 mm., over 40 per cent. being less than 1 mm. ($\frac{1}{25}$ inch) in size. At the Calumet and Hecla mills even finer material is jigged, the sands from the hydraulic separators ranging from 0.2 mm. upwards.

The jigging of such fine stuff can only result in imperfect work and large loss. It would probably prove advantageous to send all material above one millimeter ($\frac{1}{25}$ inch) to slime treatment. It is quite certain that stuff less than one-half millimeter ($\frac{1}{50}$ inch) cannot be jigged with advantage. The large loss, in the form of float copper, at the Calumet mill, is probably due mainly to the attempt to jig this fine material.

The hydraulic separators are arranged to give three or four grades of sand, according to the arrangement of the jigs in the mill. With three rows of jigs, as at the Franklin mill, 65 per cent. of the sand goes to the first, 22 per cent. to the second, and 13 per cent. to the third row. With four rows of jigs at the Atlantic mill, the distribution is 66, 21, 9, and 4 per cent. At the Osceola mill, 43.5, 31.5, 21.3, and 3.7 per cent., showing a more uniform distribution, at least for the first three spigots. In every case the last row of jigs, treating the finest sands, have but little work to do. Not only is the distribution of the sands very uneven, but the classification is almost as defective. A great deal of fine copper finds its way to the jigs, which should go to the tables, fine sands are mixed with the coarse sands, and slime, which should not appear on the jigs at all, is discharged with coarse and fine grades of sand.

The presence of fine copper, and the imperfect sorting of the sands, make it impossible to work the jigs to their full capacity without loss of copper or imperfect concentration.

By a better classification the work of jigging would be facilitated, and the capacity of the jigs largely increased, possibly even doubled. The tendency of late years has been to increase the number of jigs in the different mills, in order to diminish the losses in the form of float copper. This has been made necessary by the imperfect classification.

A better subdivision of the sands, by making the work of the jigs more uniform, would increase the capacity of the mill. This could easily be effected by varying the size and number of the openings in the bottom of the inner trough, as has already been done at the Osceola mill. Beyond this, I doubt whether the present form of separator can be much improved. It might even happen that the more uniform subdivision of the sands might make the classification less perfect, and so only aggravate the difficulty.

In order to separate effectively all slime and float copper from the sands, they should be subjected to the action of a rising current, and the sands forced to fall through and against such a rising current for at least two or three feet before being discharged. The last foot or six inches of the fall might perhaps with advantage be through clear water. The Rittinger V tubes, or "spitzlutten" apparatus, Wengler & Lowe's pointed boxes, or other "spitzkasten" with ascending current, or perhaps some modification of Professor Richards's "sorting cone," would undoubtedly do much better work than the present form of apparatus.

The radical defect of the hydraulic separator is the very short distance and limited time during which the falling sands are exposed to a rising current of water. Any float copper falling through the openings in the bottom of the inner trough has but little chance of rising again through that opening, and of being carried to the slime-tables, where it would be saved without difficulty; but must take its chance on the rough jigs, and again on the finishing jigs, thus having a double chance of being lost.

The spitzlutten apparatus is better adapted than the pointed boxes to the treatment of the coarser sands, and as it utilizes the ascending current of the slimes for separation, less clear water will be required. The pointed boxes may perhaps be used with advantage for the finer sorts of sand. The apparatus should be so proportioned as to give a velocity of current that will permit only the coarser sands to settle

in the first box. The next should have a larger cross-section, that the current may permit a finer grade to settle, and so on, the size of the channel being increased and the velocity of the current decreased in each successive box. For experiment it would be well to have the boxes made adjustable.

As an incidental result of a more perfect form of classifying apparatus, it is not unlikely that the hutchwork of some, if not all, of the roughing jigs may prove rich enough to go to the smelting works, and thus render the finishing jigs unnecessary.

Jigging.—As regards the quality of their work, the high degree of concentration, and the small amount of copper carried off in the tailings, the Collom, Scheuerman, Ball, and other jigs used at Lake Superior leave little to be desired. That this result has been accomplished in spite of the very imperfect preparation of the material in the hydraulic separator, and in spite of the almost unprecedented fineness of a large part of the material treated, reflects great credit upon the intelligence and skill of those in charge of the different mills.

The jigs and their peculiar arrangement in series have already been described quite fully in the papers already mentioned. A brief review only will be necessary here. The sands coming from each spigot of the hydraulic separator pass in succession over the sieves of two roughing jigs. The hutchwork from these goes to the finishers. On the sieves of the first of the roughing jigs a bed of coarse copper collects, through which the fine copper has to pass before reaching the hutch below. In some mills this coarse copper is automatically discharged, and the bed thus kept of uniform thickness. Generally the sieves are skimmed by hand, in some cases as often as four to six times a shift of twelve hours. This hand skimming gives a bed of coarse copper of varying thickness, from nothing to two or three inches. By the use of the automatic discharge the working of the jig is much more regular, it being possible always to have a copper bed of such thickness as will give the best results; neither so thick as to fill the jig, giving no space for the sands, and forcing them to flow over the jig half washed; nor so thin that the useful effect of a copper bed, in giving a rich hutchwork, is lost.

The second roughing jigs, over which pass the sands from the first or "head" machines, collect little or no copper, but serve to save grains of gangue, containing included particles of copper in such large proportion as to have a notable influence on their specific gravity. Such grains may contain from 6 to 30 per cent. of copper.

In some mills these rich sands are returned to the stamps. Being

already fine enough to pass through the stamp screens, but a small proportion of the grains are crushed finer, and the rest simply pass through the screens and are again caught on the sieves of the jigs. The material thus accumulates till the jig sieves are filled, and only the richer and heavier can find lodgment, the lighter and poorer stuff being carried off with the rough sands and lost.

A series of experiments at the Osceola mill showed that by this practice the copper in these sands could be brought as high as 30 or 40 per cent. The same result would have been obtained by simply allowing the sands to accumulate for a longer time on the jig sieve without removal, by which means the poorer stuff, containing less than 20 or 30 per cent., would be carried off with the waste sands, and only the richer sands would be left. As a result of these experiments it was decided that the better plan was to allow these sands to become moderately concentrated, and then take them off and send them to the smelting works as low grade mineral. A series of assays made by Mr. Patch, for the Osceola Mining Company, in December, 1877, and in March and April, 1878, showed the following percentages of copper in sands of this character, viz., 6.0, 8.62, 15.3, 18.9, 19.4, and 38.5 per cent. of copper. The first two samples were of sands too poor to be sent to the smelting works, while the last is an assay of a concentrated sample.

At the Atlantic mill the same course is pursued, and, instead of returning this material to the stamps, it is sent to the smelting works mixed with the No. 3 copper.

At the Calumet and Hecla mills all the tailings, coarse and fine sands and slimes, are collected in the tail-house, passed through a large hydraulic separator, and the coarse and fine sands jigged. These jigs yield little or no copper, but produce five or six tons a day of this rich sand, which is removed by skimming, and stamped as before mentioned.

The Hodge grinder, used at the Phœnix mill for the grinding of these rich sands, has already been described.

In view of the cost of smelting this low grade mineral, the plan of stamping or grinding these sands, and then concentrating the crushed material on jigs or tables, would seem most economical. By adding an automatic discharge to the jigs, and conveying the sands in launders to the stamps or grinding apparatus, and thus reducing the cost of handling, it is probable that poorer material, and, therefore, a much larger proportion of the rough sands, could be treated, and the copper saved. Whether sands containing less than 6 per

cent. of copper would be heavy enough to be separated by jigging remains to be proved. In any case, it is probable that enough sand of this character could be secured, even in the amygdaloid mills, to warrant the erection of a small stamp or grinding mill. In the conglomerate mills, in case it should prove advisable that all the coarser sands should be recrushed, these concentrated sands would naturally go to the fine stamps erected for that purpose.

On account of the peculiar action of the Hodge grinder in making solid pellets of the leaf copper, this machine would seem best adapted to the work of crushing these sands, provided that it can be arranged to grind the sands fine enough, and that the wear of iron be not excessive. Fine crushing, even to a slime, is essential to success in the treatment of conglomerate sands, and may prove advisable even with the amygdaloid sands.

If we examine the results of the tests of the hydraulic separator already given, we find that half the jigs, viz., those treating sands from the last two spigots, have but little work to do. In the Atlantic mill these jigs treat but four tons of rock in the twenty-four hours, while the other jigs treat about twenty-seven tons, or nearly seven times as much. At the Osceola mill two spigots give about nine tons, while the other two give twenty-seven tons, or about three times as much. At the Franklin mill, where the hydraulic separator has three spigots only, the last spigot delivers about three and a half tons, while the other two give twenty-three tons.

The amount of sand discharged on each jig could easily be equalized so as to give each its due share of the work. It is claimed, however, that these fine jigs cannot be worked to the same capacity as the jigs treating coarse material.

Two remedies suggest themselves. Our plan would be to increase the capacity of the fine jigs so that they may do as much work as the others. This would involve running the jigs at higher speed. As high as 150, 200, and even 300 strokes per minute have been successfully employed on jigs treating similar fine material. By higher speed the work done would be better; at 120 strokes the bed of the fine jigs is apt to pack; at the higher speeds this would be less likely to happen.

The other remedy would be to do away with these jigs and run the fine sands over an Evans table or other slime-washer. In the Atlantic mill this plan would require one additional table. In the Osceola mill perhaps two more tables. The jigs thus liberated

could be utilized in some other way, thus increasing the capacity of the mill.

Slime Treatment.—The slimes from the hydraulic separator are conveyed by launders to large settling tanks about 3 x 8 feet and 4 feet deep. The slimes settling in these tanks go to the Evans tables or other slime-washer. The Evans tables are used in all the mills except the Calumet and Hecla, where the Ellenbecker slime-washer is employed. At the Franklin mill the Davey slime-washer is used to concentrate the heads from the Evans table.

The Evans table has been described and figured in the papers already quoted. It is an ordinary convex revolving table, partially covered by a convex apron with a camlike outline. The table is washed clean by a strong jet of water just beyond the cam, where the apron has its smallest radius. The radius of the apron increases in the same direction as the table revolves, so that the slimes and wash-waters fall on the table nearer and nearer the outside. The charging of the slimes thus keeps pace with the flowing down of the slimes already charged, and the onflow of the wash-water keeps pace with the movement of the sands towards the circumference. The slimes flow on the table from the first half of the apron, and the wash-waters from the other half. The tails are discharged from the greater part of the circumference. Near the end of the cam the middle heads are washed off by jets of water, and finally the heads are removed by a strong jet. The spreading of the slimes and the washing are thus continuous, and as the operation on any one part of the table lasts during a whole revolution, the table can be rapidly revolved and is capable of treating large amounts of material. The middlings are sometimes elevated and go again over the same table, and sometimes collected from several tables and treated on a separate slime-washer.

A number of experiments were made at the different mills to test the working of the table, and to determine its capacity. At the Osceola mill the tables treat from 18 to 20 tons of slimes (dry weight) per 24 hours. At the Atlantic mill the slime-tables treat about 30 tons, and the sand-tables 28 to 32 tons. At the Franklin mill each table treats about 30 tons, including in this amount the retreated middlings and the tails from the Davey slime-washers.

3¾ cubic feet of slime per minute flow on each table from two ⅜-inch spigots with 4 feet head. The slime holds from 7 to 12 pounds of solid matter per cubic foot. The amount of wash-water was not

determined, but is stated by Mr. Rolker to be 2 cubic feet (17 gallons) per minute.

ASSAYS OF TAILINGS FROM EVANS TABLES.

NAME OF MILL.	NATURE OF ROCK.	CHEMIST.	PER CENT. COPPER.
Allouez mill.	Conglomerate.	Rolker, 1875.	0.90 / 0.82 / 0.78
Allouez mill.	Conglomerate.	Patch, 1876.	0 45
Osceola mill.	Conglomerate.	Patch, 1877.	0.60
Osceola mill.	Conglomerate.	Munroe, 1879.	0.525 / 0.34
Atlantic mill.	Ash-bed.	Munroe, 1879.	0.46 / 0.47
Quincy mill.	Amygdaloid.	Southworth, 1879.	0.185

It will be noticed that the assays show improvement in the working of the table, and that, even with conglomerate slimes, the tailings may be brought quite low in copper.

To test the working of the table an average sample, from a table in the Osceola mill treating conglomerate slimes, was carefully vanned, and the products of the vanning assayed, giving the following results:

	Copper.
Heads, 14 grains, @ 2.16 per cent., ...	0.30 grains.
Tails, 592 grains, @ 0.30 per cent., ...	1.76 "
Total, 606 grains, @ 0.34 per cent., ...	2.06 "

Thus the tailings contain:

Free (?) copper, ...	0.05 per cent.
Included copper, ...	0 29 "
Total, ...	0.34 "

Or 15 per cent. free (?) and 85 per cent. included. The heads from this vanning, however, as shown by the assay, are apparently not free copper, but consist of grains with enough included copper to affect their specific gravity, but not enough to allow them to remain with the heads on the table.

Another test, made on a fresh sample from the same table, taken a week later, gave:

	Copper.
Soluble in nitrate of silver, ...	0.092 per cent.
Insoluble in nitrate of silver, ...	0.433 "
Total copper in tails, ...	0.525 "

Or 18 per cent. free and 82 per cent. oxide (or included (?) copper).

A sample of the heads taken at the same time gave:

Soluble in nitrate of silver,	27.027	per cent.
Insoluble in nitrate of silver,	5.600	"
Total copper in heads,	32.627	"

On the supposition that the copper insoluble in nitrate of silver is oxide of copper, these assays show that the Evans table saves quite a large proportion of oxide of copper, and that the greater part of the loss is due to the oxide which cannot be saved. If the insoluble portion is metallic copper protected by including gangue, then the assays show that part of these grains with included copper are heavy enough, and part too light to be saved.

The Ellenbecker slime-washer used in the Calumet and Hecla mills is a small, inclined, shaking table, with end motion, discharging the heads at the upper end and the tails at the lower. Average samples of the slimes flowing on the table, and of the tailings flowing off, gave the following results:

	Per cent. copper.
Slimes, before treatment,	1.46
Tailings,	0.86
Copper saved,	0.60

Loss, 60 per cent.

95 per cent. of the slime proved fine enough to pass through a 100-mesh sieve (0.2 mm. and less). Previous tests, already described, proved that material of this fineness from the Calumet mill contains 0.57, 0.66, and 0.69 per cent. of included copper. The tailings above tested therefore contained between 0.17 and 0.29 per cent. of free copper, showing a large loss when compared with the results obtained with the Evans table. A sample of the tailings vanned for me by Mr. Ellenbecker showed free copper, very fine and difficult to save.

No test was made to determine the capacity of the tables. On the supposition that they treat all the material fine enough to pass through a 100-mesh sieve, each table would have to treat about thirteen tons (dry weight) per day. It is probable that they treat about half this, or, say six to seven tons per twenty-four hours.

The Davey tables used at the Franklin mill are similar to the Ellenbecker table, but receive an end-bump instead of a shaking motion. No tests were made of their working, as they are used simply to concentrate the heads from the Evans table, and their tailings are passed again over the latter table.

The Rittinger side-bump table was tried some years ago at the Calumet and Hecla mills, under the direction of a German expert, but failed to give satisfactory results. I was told by the superintendent that it was found difficult to wash the copper off the table. Since these experiments were made the Rittinger table has been much improved, especially in Belgium, where it is successfully used for the treatment of lead and zinc ores, replacing round tables similar to the Evans table. The Rittinger table is also successfully used at the works of the St. Joe and the Desloge Lead Companies at Bonne Terre, Mo.

The Frue vanner was tried at the Osceola mill as a supplementary machine, treating low grade "X" mineral. The results were not satisfactory, but the character of the material treated, for the most part float copper and included grains, would make it difficult or impossible to concentrate without previous fine crushing or grinding. The small capacity of this machine, six to ten tons per day, added to the high first cost, would prevent its replacing the Evans table.

All things considered the Evans table leaves little to be desired: ease of working, little or no attention being required; large product, twenty to thirty tons per day; and, finally, small losses, the loss being included copper, impossible to save by any mechanical means without finer crushing, with traces only of float copper.

Résumé.—To sum up the matter briefly, the losses in the dressing of the copper rock are chiefly due to included copper, either in metallic particles or in part as oxide of copper. The losses in the form of float copper can be, and have been in most cases, remedied by simple modifications in the arrangement of the mills, and by increasing the number of jigs and tables. The further reduction of the losses must be the subject of further investigation and experiment.

If the losses be due, in any important degree, to the formation of oxide of copper, the methods of mining must be modified to reduce this oxidation. If the loss be due, as seems more likely, principally to included copper, the remedy will lie in the fine crushing of the sands to a slime, either with or without previous concentration by jigging, and treatment of the crushed material on slime-tables.

The apparatus used, stamps, jigs, and tables, seem admirably adapted to the work, and but few changes are to be recommended. The introduction of some improved classifier, to replace the present hydraulic separator, seems imperative. By giving greater speed to the jigs treating fine sands and slime, or, better perhaps, by replac-

ing them with Evans tables, the capacity of the mills will be increased and the cost of dressing lessened. If more Evans tables be used it will be advisable to have two classes of tables,—sand-tables and slime-tables,—and to classify the slime by pointed boxes before running it on the tables.

The treatment of the intermediate products, "low grade mineral," "X," and "XX mineral," etc., has always been a difficult problem. These products are probably in great part sands rich in included copper, mixed with float copper, iron sand, and other heavy minerals. The effect of passing this material, whether coarse or fine, through a Hodge grinder or other similar machine, would probably be to release most of the included copper, roll up and condense the light float copper, and crush the iron sand and heavy minerals. The material so treated should be concentrated without difficulty.

In concluding this paper I must acknowledge, with many thanks, the co-operation and assistance of my friends of the copper region, especially Mr. Morris B. Patch, chemist to the Detroit and Lake Superior Smelting Company, Mr. William Evans, copper washer in charge of the Atlantic mill, and Mr. William O. Rowe, copper washer at the Osceola mill, who have aided me not a little in these investigations.

DISCUSSION.

PROFESSOR EGLESTON: There are a number of reasons why the losses in the Lake Superior mills are large, but they all belong to two classes. The first is defective selection of the ore, and the second, want of careful milling. The first case applies only to those mines in which the rock has to be sorted, that is, where the whole product of the mine does not go to the mill, and is generally owing to the fact that the selection is done on too large pieces, both in and out of the mine. It is, of course, expensive to bring to the surface rock which has only to go to the burrows, and consequently yields nothing, and it is still more so to send, as I have known done, such material to the mill to keep the stamps going, because the supply of pay ore on hand is not sufficient. In some of the mines pieces so large that two and sometimes only one of them will fill the skip are sent up, and material much larger left in the stopes to be block-holed below. The percentage of copper is judged of in the conglomerate mines by passing the ends of the fingers over the rock, and on the surface by the eye, which is not always

reliable, as many of the apparently barren pieces contain large pieces of copper in their interior which are often found in the dumps when they are broken. The labor expended at the surface on the rock is very large. The same amount of work could be better and less expensively done by a steam hammer, as is the case in two of the mines. This makes it possible to do away with the appreciation by touch, to break up larger pieces, and to depend more on the judgment of what can be seen. One has only to examine any burrow on the lake to see how defective the method of selection is. It would probably pay to-day to stamp most of the refuse rock of the Minnesota mine, off which a number of tributers have lived for several years, picking up pieces of copper, some of which I have known to be from one-half to twenty kilograms in weight. At present prices a good deal of the rock would pay. I have occasionally picked up pieces of from one-half to one and a half kilograms off of some of the dump-heaps of the conglomerate mines, and have seen much rich rock thrown away that might be worked with profit. This, of course, has but a relative value, and what was not true of copper at 15 cents, is true with copper at 22 cents per pound.

The chief source of loss is, however, in the dressing, and this commences at the very mouth of the mine. The shaft and rock-house are too far apart, and much of the material which would be treated if the shaft and rock houses were together is thrown away; besides, the distance between them adds to the expense, which is trifling, to be sure, if judged by the expense per ton, but which tells in the aggregate in the large mines. From the rock-house the ore goes to the stamps without any classification, and all as it comes from the shaft-house goes through the stamp-hopper. There are two capital objections to this: the first is the unnecessary beating of the copper, which in the rock itself is very thin, thus increasing the liability of loss by float, which is larger, as I have been able to prove, than is usually supposed; and the second is a decrease in stamp duty. It has been shown both with the California and the Ball stamp, as I have already had occasion to bring to the attention of the Institute, that it takes just as long to empty the mortar of material that has already passed through the screens, as it does to crush the rock and force it out. Consequently any material already of a size to pass the screens should never be put into the stamp at all. It should be sized either by screening or by the principle of the hydraulic separator, and delivered at once to the washer to which it should go.

The amount of material which would be so sized would be small in comparison to the total amount crushed in a stamp like the Ball, crushing 110 tons of rock per day; but it would cost next to nothing, and would increase the capacity of the mill just so much and decrease the losses. There are very few mills that have washers enough. After a careful study of a number of the mills I came to the conclusion that most of them could advantageously increase their number at least one-third.

The losses of copper on the screens of the washers are of two kinds: float copper and that carried off undetached from the rock, but not in sufficient quantity to change the gravity of the rock so as to prevent the stream from carrying it off. I made some experiments in catching the float copper, and was very much surprised to find how much could be caught directly from the apron of any of the washers. One has only to examine the particles with a glass to see how strongly marked the tendency to assume this condition of float is in almost any of the fine materials. I came to the conclusion that the remedy for this must be prevention by classifying the ores, so as to avoid unnecessary stamping, by an arrangement of screens to insure the most rapid delivery from them, and to have more catch-boxes. It seems to me very doubtful whether any arrangement of blankets for catching the float would ever pay for the labor. It might be found advantageous to use them as a check on the work of the mill, but it would be better to prevent its formation, than to try to catch it after it was formed. The amount carried off in the rock, and thus lost, is larger than is generally supposed. I have found in the tailings very many pieces in which a thin film of copper ran through the entire piece, and have several times made assays of tailings containing fully 50 per cent. of the entire yield of the ore; the richness depending, as I have good reason to believe, on losses of this kind. There is no way of saving this copper but by restamping it with a finer screen. This is done in some of the mills, but it is not certain that it pays. The two remedies to be used against the losses are, classification before stamping and a greater number and more care with the washers now in use.

I am not convinced that the Collom washer is the best one that can be found, but I am not prepared to propose any other. There are others and other systems that I should try, but most of the mills cannot afford to experiment even were they disposed to do so.

With regard to the losses by oxidation of which Professor Munroe speaks I am entirely skeptical. There are two theories of the

losses by oxidation: one is that oxide of copper is scattered through the ore; the other is that the fine copper becomes oxidized in contact with the water of the washer or the moisture of the mine. If either of these theories were true the whole of the oxide of copper would be lost, as a very fine powder, which would be in very small quantity, and could not be saved.

It is undoubtedly true that oxidation has taken place in the crops of all the copper deposits of Lake Superior. I have never failed to find oxide of copper in the burrows from the first lifts of the mines. In the conglomerate mines I have found both oxide and silicate, both attached to and detached from metallic copper and from each other, which were undoubtedly formed by atmospheric influences subsequent to the deposition of the copper in the rock. One of the most interesting formations I have seen in place is that of chalcotrichite at the Allouez mine, which, when first taken from the mine, is so soft that it can be easily rolled into a ball between the fingers, but which becomes so hard after a few hours' exposure that it will penetrate the fingers like a needle. When these minerals are present there is usually but little metallic copper in the rock, and in most cases none at all which can be detected by the touch. In some cases, in old burrows, I have seen beautiful needles of the carbonate. I satisfied myself, by a very careful examination of many tons of the rock, that in the lower levels oxide of copper does not occur. Occasionally a little hematite is found, and it is the red color of the powder of these very fine grains which in every case brought to my attention was mistaken for the oxide. But if the oxide was in the rock it would be extremely difficult to detect it. The question of its presence or absence was one of the problems which I attempted to solve, and after I had satisfied myself with all that inspection with powerful glasses could do, I attempted to solve it in a chemical way. The problem was, in a very large amount of rock, the total copper being only from 2 to 2½ per cent., to find a small amount of oxide of copper in the presence of a relatively very large amount of metallic copper. After making a number of trials to determine the question myself, I applied to the late Mr. Cairns to assist me. We made artificial mixtures in which we knew the amount of oxide, and we tried every known method, and among them the nitrate of silver method mentioned by Professor Munroe, but were unable to come to any satisfactory conclusion. It was very easy to determine the total copper, but not to distinguish its condition. After several months of experiment I became convinced that some of the copper in a metallic form was still

protected by rock, so as not to be affected by the reagents until after fusion. I afterwards ascertained that this was so. In making these investigations I noticed a curious fact, that wherever copper was precipitated by the battery with platinum strips in a porcelain dish the weight was always greater than the total amount of copper present whenever any of the precipitated copper fell on the porcelain. This was never the case when the whole of the copper remained attached to the platinum, or when the precipitation was done in a platinum dish. The extra weight was in this case due to oxidation, which the current, passing through the platinum dish, prevented. The oxidation seems to give a color to the theory of the oxidation of the copper in the water during the process of dressing, but the two cases are essentially different. In the laboratory precipitation the copper, in an infinite state of division, is exposed in a medium, which is more or less acid, to energetic oxidizing influences, and naturally yields to them unless a reducing current prevents it. In the other, the copper in relatively large pieces is subjected to the influence of water with no extra oxidizing influences. Under these circumstances copper does not readily oxidize, and when after very long exposure it does oxidize the thin film of oxide protects the copper below from further oxidation. It must be remembered too that the oxidation, if it takes place at all, must take place in running water, when there is a large amount of friction, so that even were the conditions favorable to oxidation the circumstances would be such as to render it almost impossible, or at least to reduce it to a minimum, so that it would be quite out of the question to account by this means for any considerable portion of the loss.

The loss which takes place is really small; but when we consider that the richest stamp-ores will not yield over 6 per cent., and that the exception is a yield as high as 3 per cent., the losses are relatively large. I have known them to be 50 per cent., and have cited instances where they were even more. In many of these cases I have been able to show that the loss was owing to the copper being covered in the gangue. I made a collection of such pieces from the tailings of one of the mills, and think there would be no difficulty in making one from any of the mills on the lake.

The real reason for want of progress in the concentration of the ores on Lake Superior is that everybody is doing the same thing, and no one is willing to take the first step in advance. Some of the mills are experimenting in a vague sort of way, but all of the small mines expect to reap the benefit without cost of the experi-

ments which the large ones ought to make, and the large ones expect the largest ones to take the first steps, and find the *paying* improvement. This is natural enough in times of depression, like that through which we have just passed, but that something can be done even by the poor mines is shown by the Atlantic mine, which, though yielding less than 1 per cent., still not only holds her own, but finds it possible even at a low price to pay a dividend.

Dr. J. A. Church: The interesting paper of Professor Munroe has shown that the defects of the Lake Superior method of dressing copper ores begin at the very first step of the process. From the stamps the ore passes to a separator, where a separation into a number of "sorts" of equal falling grains is attempted, but is very imperfectly performed. All the classes discharged from it are mixed with fine slime, which Professor Munroe has shown is found in considerable quantity, even in the coarsest jigs. Evidently some more perfect apparatus is needed to precede the entire work of separation. Professor Munroe's conclusions bear out in detail the less exact observations of the speaker, made in 1875. The faults of the Lake Superior work illustrate very well the faults most common to American practice everywhere. That is, a disposition to repeat the same process again and again on the ore, without the introduction of any difference. Perhaps the most striking example of this practice is seen in the Comstock region, where, after the ore is milled, it goes to a tailing-mill, and is treated again in precisely the same manner as in the first treatment. From the tailing-mill it goes to the gulch, where it is caught, concentrated on blankets, and milled again, by no less than six or seven men. Here is an eight- or ninefold repetition of the same process, with no change except that the little re-tailing mills in the gulch try to concentrate their pulp somewhat by running it over sluices. But their milling process is almost exactly the same as that of the large mills, and it is an odd commentary on this ridiculous system of repetition that the last re-tailing man is said to get more from his seventh reworking of the tailings than his predecessors are able to extract. A vast amount of bullion has run to waste from our silver mills, and the deficiencies of the present work show the necessity of introducing some change into the treatment at each reworking. When an ore has yielded all it can to a carefully conducted process, it is but a brutish sort of metallurgy that has no other resource than to try a series of vain repetitions on it.

Mr. Thomas Macfarlane remarked that he had, fifteen years ago, called attention to these losses, in a paper published in the *Canadian*

Naturalist, and was sorry to learn from the paper just read that so little had since been done to prevent them. He had recently assayed what was said to be an average sample of Hecla and Calumet tailings, and found in it 1.51 per cent. copper. Professor Munroe had come to the conclusion that more than one-fourth of the copper contained in the original stamp work was lost in the process of dressing, and that irrecoverably. The speaker, however, maintained that there was some possibility of saving a part of this loss with economy, and mentioned some experiments which he had made in dissolving out the metallic copper by means of aqueous ammonia. The action of this agent on fine-ground tailings, or slimes, was quite complete, all the copper being removed. On coarser material the solution of the copper was not so successful.

Dr. R. W. Raymond.—The statement made by Professor Munroe as to the defects in existing methods of ore-dressing at Lake Superior have a bearing, not only on the success of mines now working there, but also on the future availability of many beds in the copper-bearing rocks, which consist largely or wholly of finely disseminated metallic copper in sandstone. In years past I remember having seen many specimens of this kind. One was exhibited at our Cleveland meeting, from the Nonesuch mine, in which native silver also was thus disseminated. I have not lately heard of the operations of that mine. Many properties lying west of Ontonagon, in the Porcupine Mountain district, belong to the same class, so far as the fineness of the copper particles is concerned, and if they are ever to be successfully worked, it must be by the employment of efficient and systematic separating machines.

Professor Munroe: In reply to Professor Egleston's criticism I would repeat, what I have in substance already stated, that my investigations do not prove the presence of oxide of copper in the copper rock or in the waste sands, but simply indicate its presence as highly probable. Included metallic copper and free oxide of copper would exhibit similar behavior under the action of the solvents used. So far as the metallic copper is completely covered and protected by gangue it will be insoluble in nitrate of silver; if partly protected it will be partially dissolved, in this latter case acting like oxide of copper. Whether partly or entirely covered by gangue it will yield to the action of nitric acid, as the melaphyr gangue itself is soluble in that reagent, in this case again behaving like oxide.

The two tests of rock from the Atlantic mine furnish strong presumptive evidence of the presence of oxide of copper. The freshly

mined rock yielded but a trace of insoluble copper, while the rock from the stamps, presumably mined six months or a year before, showed that nearly one-third of its copper had become insoluble. This change in the solubility of the copper can only be accounted for by supposing the formation of oxide. On the other hand the copper rock is not uniform, neither in the amount of copper it contains, nor in the manner in which the copper is distributed through the rock. It is possible that the sample of freshly mined rock contained little or no finely disseminated copper, and that the sample from the stamps contained an unusually large proportion of included copper. While this does not seem probable, in view of the fact that great care was taken to secure an average sample in each case, still it shows that it is not safe to generalize from so few assays, and while the presence and the formation of oxide of copper in the copper rock seems extremely probable, it cannot be claimed as proved. That any oxide of copper is formed during the washing process is, as Professor Egleston states, very doubtful; if formed at all, it must be before milling, in the mine, or on the dump.

To prove the formation of oxide of copper in the mine, further tests are required. The effect of finer comminution should be tried, the sample supposed to contain oxide of copper being alternately treated with nitrate of silver and ground in a mortar. If the amount of copper dissolved by nitrate of silver increases with the fineness of the sample, and if the whole of the copper can be brought into solution in this way, the absence of oxide of copper is proved, and *vice versa*. To test the effect of exposure to mine gases, etc., a large number of assays should be made on samples of freshly mined rock, etc., in order to eliminate the variations due to the heterogeneous character of the copper rock.

The advisability of using revolving screens or some form of sizing apparatus has been suggested in the discussion. I do not consider this necessary. The separation of metallic copper from its gangue is a comparatively simple and easy ore-dressing problem, owing to the great difference in specific gravities. A very close classification of the sands is not required, especially where the English method, of jigging through a bed, is used. While the use of screens would probably increase the efficiency of the mill by increasing the richness of the product, and diminishing the number of jigs required, the cost of the screens would be large and the expense of repairs and maintenance heavy, and not warranted by the gain which would result. The main point in which improvement is necessary is in a better separation of the slimes from the sands,

that the very fine float copper may not go to the jigs, where it is lost, but to the tables, where it may be saved. This can be accomplished, as already pointed out, in a much simpler way than by the introduction of an elaborate screening plant.

I do not agree with Professor Egleston in his statement that few of the mills have washers enough for the work. On the contrary, it has already been shown that if the sands were to be more uniformly distributed, the work would probably be equally well done with about two-thirds the present number of machines. There is no doubt, however, that it would in most cases be well to increase the number of slime-tables.

REMARKS ON A GOLD SPECIMEN FROM CALIFORNIA.

BY PROFESSOR GEORGE W. MAYNARD, NEW YORK CITY.

In the course of an examination of some of the California hydraulic mines in November last, I visited the property of the Gold Run Ditch and Mining Company, near Dutch Flat, Placer County. This is one of the most important and extensively worked mines in the district. The gold occurs in the characteristic blue gravel, which averages 175 feet in thickness, the lower 40–60 feet being made up of cement gravel, which resists the action of water, and can only be loosened by blasting. The specimen which I have the pleasure of laying before the members was presented to me by Mr. I. L. Gould, the intelligent superintendent of the mines, who informs me that he found it in the old river channel, about twelve feet above the bedrock. The flattened water-worn nugget of gold, about the size of a kidney-bean, is imbedded in blue cement gravel, around which quartz has crystallized. When found the projecting portion of the nugget was enveloped in quartz, the core of gravel and gold having only been discovered after breaking the lump of quartz. On the quartz there is a shell of cement.

The specimen is a remarkable confirmation of the theory of the deposition of quartz from solution.

DISCUSSION.

Dr. T. Sterry Hunt, in commenting upon this occurrence, remarked that it is in accordance with what we already know of the recency of some of the quartz of this region, and cited the microscopic

studies of John Arthur Phillips, who has shown that a great part of the siliceous deposit from certain thermal waters of Lake County, California, and from the Steamboat Springs of Washoe County, Nevada, is of the nature of crystalline quartz. Dr. Hunt then gave an account of some observations made by him at the Blue Tent placer mine, in Nevada County, California, in 1877, showing that the process of depositing quartz is there going on in the auriferous gravel of the region, independent of thermal waters, and is connected with the sub-aerial decay of the silicates in the gravel, which is here made up in great part of the débris of the crystalline Huronian schists of the region, including much greenstone or diorite rock. The gravel below the drainage-level is greenish or bluish in color, and contains disseminated pyrites, together with trunks of trees in the condition of lignite, while the feldspar and hornblende of the greenstone are undecayed. Above the drainage-level, however, these silicates are more or less decomposed, the greenstone pebbles becoming earthy in texture, rusty in color, and exfoliating, and the accompanying pyrites oxidized, while the lignite is more or less completely silicified, being sometimes converted into agatized masses, often with drusy cavities lined with quartz crystals, and at other times only penetrated or injected with siliceous matter which has filled the pores of the exogenous wood, the vegetable tissue of which still remains, often incrusted with crystals of quartz. In still other cases, a slow subsequent decay of the latter, in coniferous woods, has left these siliceous casts in the form of bundles of fibres, which have been mistaken for asbestos. The various specimens from this locality illustrate perfectly the theory of silicification of vegetable structures, set forth by the speaker in 1864,* based on his own microscopic studies conjoined with those of Göppert and of Dawson.

The silica by which the tissues are thus successively filled and replaced is, according to the speaker, that which is set free in a soluble form in the decay of the silicates in the gravel. The lignite in the undecomposed and unoxidized portions of this which lie below drainage-level is, as yet, unsilicified. Dr. Hunt acknowledged his obligations to Mr. D. T. Hughes, a member of the Institute, in charge of the mine in question, and a skilled and careful observer, who had called his attention to the facts just set forth.

Professor Egleston: The specimen which Mr. Maynard ex-

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* See Canadian Naturalist, N. S., vol. i., p. 46; also Hunt's Chem. and Geol. Essays, p. 286.

hibits I consider one of the most remarkable that has ever been brought to the attention of the Institute, since it clearly proves a number of very interesting points with regard to the formation of gold in alluvial deposits.

The usual theory is that placer deposits, whether shallow or deep, have been formed by the destruction of the gold rock previously existing in veins. This theory has always seemed to me to be untenable for several reasons, and this specimen proves it to be so.

The first of these is the shape of the nuggets and of the grains which are formed in the alluvial sands. They are either flat with rounded edges and convex surfaces, or very irregular in shape with numerous concave surfaces. A nugget rounded like a water-worn pebble is an exception. If they had resulted from the grinding up or destruction of vein matter they could not have had the irregular forms which they usually present. As gold is quite soft, it must have been pounded flat or ground to an impalpable powder by any force sufficient to break up the vein matter and detach the metallic particles from their gangue. The abrasion would have produced leaf gold so fine that little of it would have been concentrated by any natural action in any one place in sufficient quantities to detect it. It would either have been lost by being disseminated in very small quantities, as in the Philadelphia clays, or have entered into solution and passed into the sea.

In addition, it is difficult after examining the models of the historical gold nuggets in the mineralogical collection of the School of Mines—one of which, from Ballarat, weighed 2166 ounces, another from Melbourne 1740 ounces, and another from Siberia 1200 ounces—and observing their extremely irregular forms, to believe that such heavy masses could have ever been so thoroughly detached from their vein matter, and have been carried any distance by water, without having an entirely different form from that they now have. The only way in which their irregular form can be accounted for is by the supposition that they were formed by constant deposition around a small nucleus, which deposition continued for a very long time.

The constitution of the gold in the nuggets is generally such as to preclude the supposition of its having been accumulated from the destruction of vein matter. Vein gold generally contains a considerable quantity of foreign metal, which makes it more or less low grade, and often a certain part of it is rusty and difficult to amalgamate. This variation in quality is shown by the analysis of nearly all vein gold. Rustiness has been supposed by some to be entirely

superficial, but I have reason to believe that though it is often so, it is owing in many cases, and perhaps in the majority of them, to the chemical composition of the metal. I have recently called the attention of the authorities in Washington to this subject, and have been promised their assistance in investigating it.

Gold is generally considered as a metal soluble only with great difficulty. Some recent and extended experiments, on this subject, which are not yet completed, but which I am still pursuing, prove the contrary to be the case.

It was well known, even before the researches of Sonstadt, that gold was slightly soluble in alkaline solutions, and that from these solutions it is precipitated by organic matter. Some metallurgical processes have been founded upon this principle. A charcoal filter is now used to precipitate gold from its solution on a large scale. Mr. Newberry, the Australian geologist, published some years ago a paper relating to this subject.

The conditions under which the solution takes place in the laboratory experiments are just those which are most frequently found in nature, especially in the deep placer mines of California, where the waters of filtration either contain or absorb from the rocks all the material necessary for the solution of the gold. Independently of the solution of gold in alkalies and the well-known deposition of gold and quartz at the same time from the decomposition of these alkaline solutions by organic acids in the soil, and of the well-known cases of solution of gold in iodine and bromine and salts containing them, there are other agencies which render the metal much more easily soluble. The presence of nitrates and chlorides in the soil is not uncommon. Few surface waters are free from chlorides, and it requires but a very small proportion of nitric acid in a water in connection with them to take up the metal. I have proved recently that the presence of a trace is quite sufficient in a dilute solution at ordinary temperatures and pressure to produce a solution, after an exposure of three months, which is appreciably colored.

We know from Sonstadt's researches that sea-water contains nine-tenths of a grain of gold to every ton of water. This quantity is exceedingly small, but time and quantity are things in which nature is bountiful, and we have only to suppose that the waters, containing a very small amount of gold in solution, filtered an indefinite time through ground containing organic or any other material that would precipitate the gold in the metallic state, to account for the deposition of the metallic gold frequently found in pyrites. Such a

method of precipitation would fully account for the richness of the deep placers on the bed-rock and their well-known poverty on the surface; for the waters penetrating through at the surface would pass rapidly and leave but a small amount, but being confined by the impenetrability of the bed-rock they would be retained a much longer time there, and the gold would be deposited from them.

The specimen presented by Mr. Maynard shows the gold deposited in just such a nucleus. This nugget is of the same character as those generally found in placer deposits. It would have been impossible for the gold to have got into this nugget in any other way than by solution; and it seems quite certain that the iron of the blue gravel was the precipitating cause in the first instance,—the first deposit once formed the metallic gold decomposed the solution, and its own weight was increased,—and that the precipitation of the quartz was contemporaneous with it.

Many a photographer in the days when photographs were fixed with gold, has found his gold solution freed from gold, it having been precipitated by very small particles, either of metallic or of organic matter, which had accidentally fallen into it, in the form of a small nugget at the bottom of his liquid, in all respects similar to the one found in this specimen.

I have been able to prove that different organic materials act differently. Petroleum covering a weak solution of gold precipitates it after months of exposure in fine crystals, that can at first only be seen in the liquid by holding it against the sun, but which increase gradually in size, and fall to the bottom. Materials with an organic structure, such as leather and cork, precipitate the gold in their interior, and become themselves transformed into the metal. In this connection I may state that I have recently examined a pseudomorph of chalcocite after wood, from Texas, very rich in gold, in which the transposition was evidently from a sulphate of copper, the organic matter of the wood reducing the gold to a metallic state, and at the same time the sulphate to a sulphide. Near the same spot was the trunk of a tree, entirely transformed into oxide of iron from the decomposition of a sulphate to a sulphide, and subsequent oxidation of the iron. The specimen was not sent to me, but I have little doubt that, like the copper, it contained gold.

Every one who has examined the gold extracted from different placer deposits, whether shallow or deep, must have been struck by the great difference, not only in the size, but in the shape of the grains which are obtained from them, and these seem to show that

in different localities the causes of solution might have been different, and the material which produced the precipitation not the same.

It is a remarkable fact that much of the gold of Brazil is found in connection with specular iron and limonite, and that the black sands of California not only contain it, but many other metals. It is significant that crystals are rarely found in these sands, which fact has hitherto been accounted for by the supposed destruction of the rock, but seems to me to be more satisfactorily accounted for by the rapidity of the action in placers, the loose gravel permitting a flow with comparatively few obstacles, in contradistinction to the slowness of vein deposits, where, the waters flowing with more difficulty, the gold has time to be deposited as crystals. The study of these gold crystals, whether macro- or microscopic, has been a delightful source of recreation to me for many years.

PROFESSOR W. C. KERR: With regard to the point last discussed, in connection with the subject of this quartz-enveloped, water-worn nugget, I presume some of the members have seen the statements and discussions of Professor Lieber, the geologist of South Carolina, in his reports, published more than twenty years ago, in reference to the collection, the aggregation of gold into visible particles, coarse gold, and nuggets, in the tailings of the gravel mines of the Carolinas. But if not, I will state that no fact about these gold deposits is more commonly known or more universally acted upon than this; and these gravels are regularly reworked every ten or twelve years, the second and third crops often equalling the first; and my friend, Major Hotchkiss, near me, announces that the same is true in Virginia. It is true throughout the gold belt of the South. Now as to the agency by which this aggregation of the fine, diffused, invisible gold is effected, no doubt Professor Egleston, who has just taken his seat, has indicated correctly the direction in which the solution of the problem is to be sought, viz., the gold-dissolving power of alkaline waters. And Dr. Hunt has just shown us where to look for such solvents in these gravels, and also the source of the silica. In fact, the two processes, the solution and aggregation of the gold and the deposition of the quartz, are parts of the same complex chemical action which is going on continually in all these superficial gold gravels, and the two proceed *pari passu,* and are complementary to each other. You may almost see these processes going on under your eyes. These deposits are simply the half decayed débris of the feldspathic gneisses and schists of the neighboring hills and slopes. The feldspathic sands are undergoing kaoliniza-

tion by the percolation of meteoric waters, the resulting alkaline solutions, passing downwards, meet, dissolve, and redeposit the gold, while the liberated silica is precipitated, often attaching itself to and replacing any organic substances that may be present, thus furnishing an explanation of the extensive silicifications occurring in these deposits. I have often seen entire trunks of trees petrified in this way, lying within a few feet of the surface. I had in my hand last week a section of such a tree bole not less than a foot thick, in the middle of which was a rift, representing a crack in the log, the walls of which were faced with the projecting pyramidal terminations of perfectly formed minute crystals of quartz. Thus, as I said, you may almost see, with your own eyes, these two concurrent, mutually interdependent chemical processes,—the reaccumulation of workable gold in old, exhausted washings and the deposition of the residual silica. And why should we not expect to find these world-old forces of mineral accumulation, transference, and deposition in activity now as well as at any former period?

THE MICA VEINS OF NORTH CAROLINA.

BY W. C. KERR, STATE GEOLOGIST, RALEIGH, NORTH CAROLINA.

A BRIEF sketch only is here intended, with a few illustrations, in order to give a general notion of the character and structure of these veins. I have stated elsewhere, several years ago, that these veins were wrought on a large scale and for many ages by some ancient peoples, most probably the so-called Mound Builders; although they built no mounds here, and have left no signs of any permanent habitation. They opened and worked a great many veins down to or near water-level; that is, as far as the action of atmospheric chemistry had softened the rock so that it was workable without metal tools, of the use of which no signs are apparent. Many of the largest and most profitable of the mines of the present day are simply the ancient Mound Builders' mines reopened and pushed into the hard undecomposed granite by powder and steel. Blocks of mica have often been found half imbedded in the face of the vein, with the tool-marks about it, showing the exact limit of the efficiency of those prehistoric mechanical appliances. As to the geological relations of these veins, they are found in the gneisses

and schists of the Archæan horizons, in that subdivision which I have provisionally classed as Upper Laurentian, the Montalban of Dr. Hunt. These rocks are of very extensive occurrence in North Carolina, constituting in fact the great body of the rocks throughout the whole length of the State,—some 400 miles east and west,—being partially covered up, and interrupted here and there by belts of later formation. Mica veins are found here, in fact may be said to characterize this horizon everywhere, from its eastern outcrop, near the seaboard, to and quite under the flanks of the Smoky Mountains. It is, however, in the great plateau of the west, between the Blue Ridge and the Smoky, that the mica veins reach their greatest development, and have given rise to a very new and profitable industry,—new and at the same time very old.

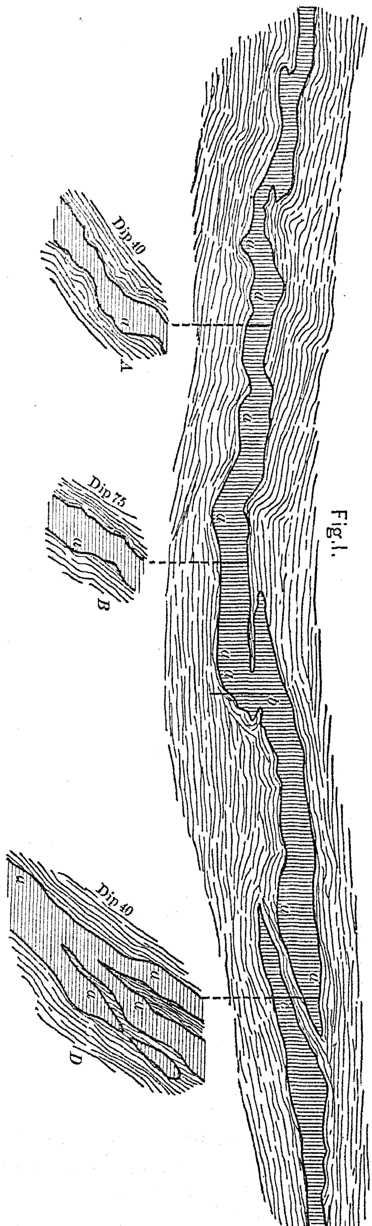


Fig. 1.

It may be stated as a very general, almost universal, fact, that the mica vein is a *bedded* vein. Its position (as to strike and dip) is dependent on and controlled by, and quite nearly conformable to, that of the rocks in which it occurs, and hence, as well as on account of their great size, some observers, accustomed to the study of veins and dikes and the characters of intrusive rocks in other regions, have been disposed to question the vein character of these masses at first. But a good exposure of a single one of them is generally sufficient to remove all

doubt on this score. The mica vein is simply and always a dike of *very coarse granite*. It is of any size and shape, from a few inches—generally a few feet—to many rods (in some cases several hundred feet) in thickness, and in length from a few rods to many hundred yards, extending in some cases to half a mile and more. The strike, like that of the inclosing rocks, is generally northeast, and the dip southeast, at a pretty high angle; but they are subject, in these respects, to many and great local variations, all the conditions being occasionally changed, or even reversed. An idea may be formed of the coarseness of these veins from this statement, that the masses of cleavable feldspar and of quartz (limpid, pale yellow, brown, or, more generally, slightly smoky), and of mica, are often found to measure several yards in two or three of their dimensions, and weighing several tons. I have a feldspar *crystal* from one of these mines of nearly a thousand pounds weight, and I have known a single block of mica to make two full two-horse wagon-loads, and sheets of mica are sometimes obtained that measure three and four feet in diameter.

There are many peculiarities about these veins. Among the most important, in a practical sense, is the arrangement of the different constituents of the vein *inter se*. Sometimes the mica, for example, will be found hugging the hanging-wall; sometimes it is found against both walls; again it may be distributed pretty equally through the whole mass of the vein; sometimes, again, it will be found collected in the middle of the vein; in other cases, where the vein varies in thickness along its course, the mica will be found in bunches in the ampullations, or bellies of the vein; in still other cases, where the vein has many vertical embranchments, the mica will be found accumulated in nests along the upper faces of these processes or offshoots. These features of structure will be best understood from a few representative diagrams.

Figure 1 is a horizontal section, with several transverse vertical sections, of a typical vein in Yancey County, at the Presnel mine. The length of the section, *i. e.*, of the portion of the vein that has been stripped, is 125 feet; the thickness varies from 3 to 10 feet, except at a few points, as *b*, *c*, where it is nearly 20 feet.

The crystals of mica are found in this mine generally near the foot-wall, in the recesses or pouches; at *c*, however, as seen in section D, it is found next the hanging-wall.

The inclosing rock in this case is a hard, gray slaty to schistose gneiss. The relation of the vein to the topography is seen in Fig. 2.

Another characteristic vein is well exposed at the Point Pezzle mine, in the same county, which has been wrought very successfully for several years. (Fig. 3.)

The inclosing rock is the same as in the former case. The mica here is found mostly next the hanging-wall, and also in the offshoots or branches of the vein, as shown in the vertical section at *d*. This vein illustrates the exceeding irregularities which are often found in these intrusive masses,—irregularities in form, size, and position,—

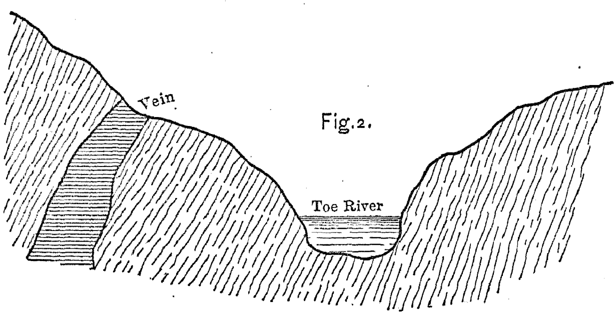


Fig. 2.

and the force with which the inclosing rocks have been crowded and bent and split in the effort of the vein matter to insert itself. This vein is 40 feet thick at *b*, and 1 to 2 feet at *c*.

Figure 4 furnishes another illustration of the same points.

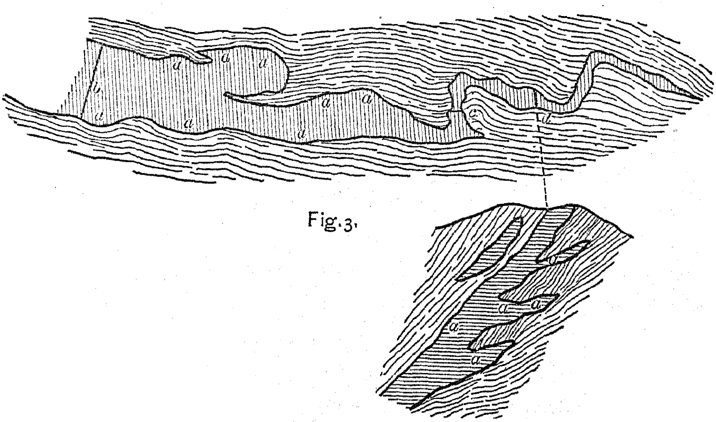
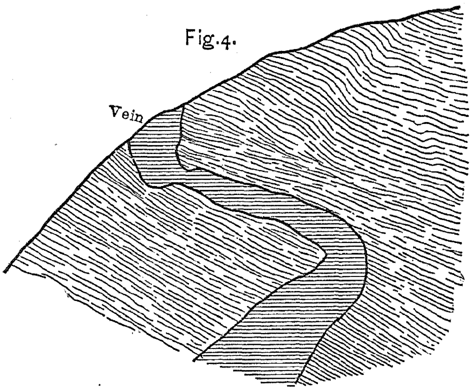


Fig. 3.

Another class of veins is represented by the accompanying cross-section of one of the largest and most productive veins, on which was opened the first mica mine in this region (Sink Hole), in the bottom

of an old pit, partly filled up and overgrown, bottom and top, with heavy forest trees of the age of three hundred to four hundred years. (Fig. 5.)

Fig. 4.



The lower part of the diagram represents an enlarged cross-section of the vein at *b*. The wall-rock, 1, 1, is a soft, decomposed micaschist. A horse, of this wall-rock, is imbedded in the body of the vein on both sides, 1, 1. At 3, 3, occur interpolations of smoky quartz, and next to this, on both sides, at 4, 4, most of the best mica is found, although occasional masses of marketable mineral are found throughout the vein, at 5, and 2, 2. In the side vein, however (*a*), the mica is found mainly in the middle.

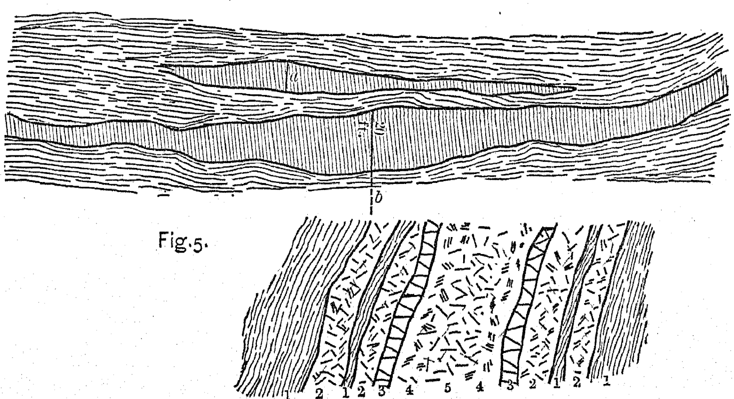


Fig. 5.

The extent and value of the mica industry may be indicated by some statistics of this one mine. The vein has been worked out to water-level for nearly half a mile, and it is estimated that the

aggregate length of its tunnels is more than six miles, and the yield of marketable mica above 40,000 pounds.

In preparing the blocks of mica, splitting and cutting to the forms and sizes demanded by the markets, there is a waste of nine-tenths to nineteen-twentieths, even in a good mine.

The feldspar, which constitutes the larger part of the mass of these veins, is often found converted into beds of the finest kaolin; and, curiously enough, this was one of the first and most valuable exports to England in the early part of the seventeenth century, "packed" by the Indians out of the Unaka (Smoky) Mountains, and sold under the name "unakeh" (white). This kaolin, like the mica, will doubtless soon come again into demand, after lying forgotten for generations.

These are only a few of the more prominent characteristics of these very interesting veins. I have not referred to their singular richness in rare minerals, as samarskite, uraninite, gummite, allanite, etc., nor to many curious and unexplained relations between the marketable character of the mica,—size, color, purity, fissility, etc.,—and the special matrix in which the blocks are imbedded. I do not know a better region for the study of the structure and origin of veins in general.

THE GOLD GRAVELS OF NORTH CAROLINA—THEIR STRUCTURE AND ORIGIN.

BY W. C. KERR, STATE GEOLOGIST, RALEIGH, NORTH CAROLINA.

When Agassiz and his party of geologists commenced their exploration of the interior of Brazil and the Amazon region, one of the first and, to the last, one of the most novel and striking phenomena which met them everywhere was the great depth of decomposed or partially decayed rock *in situ*, which mantles, and for the most part conceals, the underlying strata. The same facts strike all geological observers from the North who happen to penetrate the middle and southern latitudes of the Atlantic States. In North Carolina, *e. g.*, the entire middle and western regions, outside of the Quaternary clays, sands, and gravels of the East,—that is, all that portion of the State occupied by the Archæan and Mesozoic rocks,—show everywhere this peculiarity, so new to those accustomed to glaciated surfaces. Not only do the hills and slopes, the mountain chains and

spurs, present everywhere to the eye this superficial covering, but even the more level tracts and the valleys. The railroad cuts give very good exposures of this covering, and furnish, everywhere, abundant opportunities for the study of its structure and history. Some of the more obtrusive facts are these: the thickness of this covering varies from a few feet to 30 or 40, and often 60 and 75, and even 100 feet, and bears an obvious relation to the character of the underlying rock, being least where this is most refractory, and *vice versa;* the rock is generally nearest the surface in the crests of the hills. The upper portion of this earthy envelop for several feet beneath the soil is homogeneous and structureless; but lines of structure soon make their appearance, becoming more pronounced with the depth. These lines of structure are commonly coincident with bands and ribbons of differently colored earths, which, on closer inspection, show differences in their materials also, these differences becoming more and more strongly marked as they are traced downward, until they pass by insensible gradations into the solid rock beneath. The obvious and necessary conclusion from these observations gives itself, viz., that the rocks of the region are and have long been undergoing a slow chemical decomposition and disintegration from the action of atmospheric forces, this decay being too rapid, however, to be overtaken by the abrasive and transporting power of these same agencies.

So far the general and obvious facts, plain to be read by the man that runs. A little closer inspection reveals another set of facts. It is easily discovered that these mantles of earth and half-decayed rocks are not strictly *in situ*, but have been subjected to some sort and degree of movement, and that the materials have undergone at least a

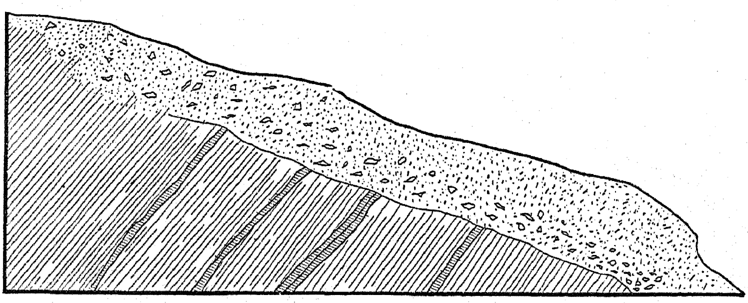


Fig. I.

partial rearrangement in certain situations and under certain conditions. In general on the summits of the hills there has been no change, but

descending the slope, however gentle, a tendency to a sorting and arrangement of materials appears, and this becomes more observable with the distance. At first the fragments of quartz and other hard rocks are sharply angular, and are distributed equally and irregularly through the mass, or in lines corresponding to the bedding of the rocks. Descending a few rods the rock fragments have "settled" somewhat; they are found more thickly strewn towards the bottom, and are less angular. Descending still further all the coarser fragments are found accumulated in a layer of cobbles or pebbles, with only the interstices filled with earth and gravel.

Combining sections of this covering taken from different points, from the hilltop to the bottom of the slope, which commonly terminates in a ravine or valley, or the bed of a stream, we have the appearance shown in Figure 1.

The obvious interpretation of these facts is that there has been a movement or flux of the earthy mass in the direction of the slope. And this notion is confirmed by an occasional observation which is represented in Figure 2.

The difficulty at once arises how to account for a flow of such materials with such results. The ordinary action of flowing water is, of course, excluded. The mere action of gravitation will not account for the phenomena,—slipping or sliding down hill. This, doubtless, often happens on very steep declivities, but such cases are quite exceptional and are easily distinguished. The movements we are considering have taken place at every degree of inclination, from one degree and less upwards, and occasionally on a level, *or even up hill.*

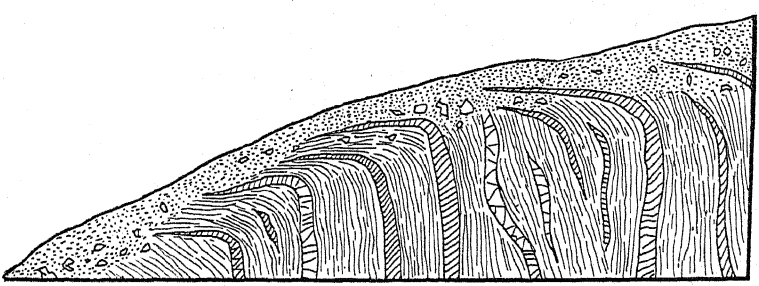


FIG. 2.

After puzzling over these phenomena for half a dozen years, and wondering that there is no explanation in the books, or even any discussion of the subject or note of the facts, not even in Ger-

kie's *Great Ice Age*, it occurred to me that the only possible solution must be sought in the action of frost. The alternate freezing and thawing of such a mass of earth must needs produce just the effects we have been considering. The earth, saturated with water, in the process of consolidation under the action of cold would, of course, expand just as if it were all water, and in thawing there would be a slight movement of the parts and particles of the mass *inter se*, and of course a settling of the heavier fragments; in other words, the movement would be the same in kind (though not in amount) as that of a glacier. These masses may be considered earth glaciers, and I have ventured to denominate this group of phenomena, and these peculiar superficial accumulations, *frost drift*. Now the ordinary glacial phenomena are wanting in North Carolina, with, perhaps, the exception of a few morainal ridges in the gorges of the higher mountains. But during the glacial period, of course, the cold must have been intense enough to account for the depth and extent of action which the theory of *frost drift* supposes.

I was led to these results from the particular study of the gold deposits of the State. They have all been formed in this way. There are probably five hundred square miles of gold drifts in North Carolina. They are found through a range of four hundred miles east and west, from the lower waters of the Roanoke, near Weldon,

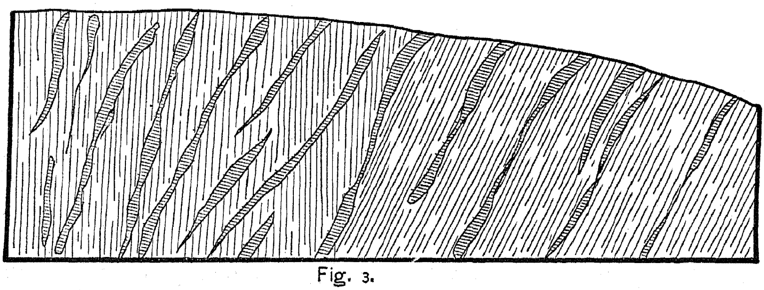


Fig. 3.

to the extreme western border, the county of Cherokee, and they belong to all the different subdivisions of the Archæan rocks of the State. The two most extensive deposits are found in the middle region, on the Yadkin and Catawba rivers, among the low ranges and spurs of the mountains. The schistose and slaty rocks, highly inclined and much contorted and dislocated, are in many places penetrated by innumerable small veins and seams of gold-bearing quartz. (See Fig. 3.) In the disintegration and breaking down of these rocks, and the movements of the débris in the manner described, it

is evident that the gold particles, with the heavier crystalline minerals, will be found accumulated near the bottom of the drifts, on or near the surface of the bed-rock, or "slate," as the miners call it. The gold mining of modern times began sixty years ago in this region from the accidental discovery of a twenty-eight pound nugget by a boy in one of the streams of this region. Most of the simple and effective appliances now in use everywhere for the separation of gold from such deposits—the long tom, the sluice, the riffle-box, etc.—were devised and used in this region, and were carried hence to California when, twenty-five years later, the trained miners of this region emigrated in a body to that newer and richer field. Since that emigration there has been but little placer mining done in North Carolina. Still this sort of mining has never entirely ceased, and in some sections, and by a few families, it has been followed continuously to the present. The richest deposits within reach of water have been worked over, but there are large areas still untouched, because inaccessible to water without considerable outlay for ditching, canalling, and fluming, to which neither the capital nor the enterprise of the region is equal.

SUPPLEMENT I. TO A CATALOGUE OF OFFICIAL REPORTS UPON GEOLOGICAL SURVEYS OF THE UNITED STATES AND TERRITORIES, AND OF BRITISH NORTH AMERICA.

BY FREDERICK PRIME, JR., LATE ASSISTANT GEOLOGIST OF PENNSYLVANIA.

IN this supplementary list no titles to which an * is prefixed have been seen by the compiler; and he will be most thankful to have any omissions or inaccuracies in the list sent to him to be published as a second supplement in the next volume of the *Transactions*. The author is indebted to Messrs. Robert Clarke, of Cincinnati; Thomas Macfarland, of Towanda; Professor F. W. Simonds, of Chapel Hill; and Professor J. J. Stevenson, of New York, for valuable assistance.

CEDAR POINT IRON COMPANY, BALTIMORE,
July 31st, 1880.

ALABAMA.

Geological Survey of Alabama. Report of Progress for 1877 and 1878; by Eugene A. Smith, State Geologist. Montgomery, 1879. 8vo., 139 pp., and 4 maps.

Alaska.

Report upon the Customs District, Public Service, and Resources of Alaska Territory; by William Gouverneur Morris, Special Agent of the Treasury Department. Washington, 1879. 8vo., 163 pp., and 11 plates.

British America.

Hind's Report on the Assiniboine Expedition has the title "Report of Progress, together with a Preliminary and General Report on the Assiniboine and Saskatchewan Exploring Expedition; by Henry Youle Hind." Toronto, 1859. 4to., 201 pp., 6 maps, 2 sections, and 3 plates.

*Papers relative to the Exploration of the Country between Lake Superior and the Red River Settlement; by S. J. Dawson and Henry Y. Hind. London, 1859. Folio, -- pp., and 4 maps.

California.

*Williamson and Derby. Report on the Topography of California, 1850.

*Randall. Report of Special Committee in favor of a Geological Survey of California, 1851.

*Report of Hon. T. Butler King on California. Washington, 1850. 8vo., 72 pp.

Trask's Geology and Industrial Resources of California. Baltimore, 1851, contains 3 and not 4 maps.

Trask's Geology of the Coast Mountains. 1855. Has the title "Report on the Geology of the Coast Mountains, embracing their Agricultural Resources and Mining Productions; also portions of the Middle and Northern Mining Districts; by John B. Trask." ———, 1855. 8vo., 95 pp.

Trask's Geology of Northern and Southern California. Sacramento, 1856, contains 66 pp.

Annual Report of the State Geologist for 1863 was printed at Sacramento, 1864.

Geological Survey of California. J. D. Whitney, State Geologist. Map of a portion of the Sierra Nevada adjacent to the Yosemite Valley, from survey made by Charles T. Hoffmann and J. T. Gardiner, 1863–1867. Scale, 2 miles to 1 inch.

The following corrections should be made in regard to the works on the Yosemite Valley:

The Yosemite Book. A Description of the Yosemite Valley and the adjacent Region of the Sierra Nevada, and of the Big Trees of California. New York, 1868. 4to., 116 pp., 2 maps, and 28 photographs. [J. D. Whitney.]

[Only 250 copies published.]

The Yosemite Guide-book. Cambridge, 1869. Small 4to., 155 pp., and 2 maps. [J. D. Whitney.]

[Second Edition. Cambridge, 1871. 16mo., vii and 133 pp., and 2 maps.]

[Third Edition. Revised and corrected. Cambridge, 1874. 16mo., vii and 186 pp., and 4 maps.]

CANADA.

*Report of the Commissioners for Exploring the Country lying between the Rivers Saguenay, St. Maurice, and St. Lawrence. ———, 1831. 12mo., 47 pp. [Andrew and David Stuart.]

The following five titles are corrections of the list published in the catalogue:

Geological Survey of Canada. Report of Progress for the year 1844. Montreal, 1846. 8vo., 110 pp. [W. E. Logan.]

Geological Survey of Canada. Report of Progress for the year 1845–6. Montreal, 1847. 8vo., 125 pp. [W. E. Logan.]

Geological Survey of Canada. Report of Progress for the year 1846–7. Montreal, 1847. 8vo., 66 pp. [W. E. Logan.]

Geological Survey of Canada. Report of Progress for the year 1847–48. Montreal, 1849. 8vo., 165 pp. [W. E. Logan.]

Geological Survey of Canada. Report of Progress for the year 1852–53. Quebec, 1854. 8vo., 179 pp. [W. E. Logan.]

*Canadian Pacific Railway. Report of Progress of the Explorations and Surveys up to January, 1874. Ottawa, 1874. 8vo., xiii and 286 pp., map, and 9 diagrams.

Reports of Mr. A. Michel and Dr. T. Sterry Hunt, on the Gold Region of Canada. Transmitted by Sir W. E. Logan to the Hon. Commissioners of Crown Lands, February 14th, 1866. Montreal, 1866. 8vo., 72 pp.

List of Publications of the Geological Survey of Canada. Museum and Library, 76 St. Gabriel Street, Montreal. Alfred R. C. Selwyn, Director. Montreal, 1873. 8vo., 7 pp.

Geological Survey of Canada. Alfred R. C. Selwyn, Director. Report of Progress for 1877–78. Montreal, 1879. 8vo., xx, 15, 187, 37, 31, 13, 34, 6, 36, 32, 52, 14 pp., 4 maps, and 17 plates.

FLORIDA.

*Florida: its Climate, Soil, and Productions: with a sketch of its History, Natural Features, and Social Condition. Prepared officially by J. S. Adams, Comm. of Immigration. Jacksonville, 1869. 8vo., 151 pp.

INDIANA.

Eighth, Ninth, and Tenth Annual Reports of the Geological Survey of Indiana, made during the years 1876–77–78; by E. T. Cox, State Geologist. Indianapolis, 1879. 8vo., 541 pp., and 4 maps.

IOWA, WISCONSIN, AND ILLINOIS.

Owen's Report contains 25 plates.

KENTUCKY.

Owen's First Report contains 9 maps.

LOUISIANA.

*An Account of the Red River in Louisiana, drawn up from the Returns of Messrs. Freeman and Custis to the War Office of the United States, who explored the same in the year 1806, n. p. n. d. 8vo., 63 pp., and table.

MASSACHUSETTS.

Commonwealth of Massachusetts. House Bill No. 266. Boston, 1874. 8vo., 4 pp.

Report of the State Board of Education on the Proposed Survey of the Commonwealth. Boston, 1874. 8vo., 19 pp.

MICHIGAN.

Houghton's Third Report contains a map.

*Senate Ex. Doc. No. 175, Special Session. A Report of John Stockton, Superintendent of the Mineral Lands on Lake Superior. March 19th, 1845. Washington, 1845. 8vo., 22 pp., and map.

MINNESOTA.

Geological and Natural History Survey of Minnesota. Circulars Nos. 1, 2, 3, and 4.

MISCELLANEOUS.

Schoolcraft's Report, also printed "Summary Narrative of an Exploratory Expedition to the Sources of the Mississippi River in 1820; resumed and completed by the Discovery of its Origin in Itasca Lake in 1832;" by Harry R. Schoolcraft. Philadelphia, 1852. 8vo., xx and 596 pp.

*XXXIXth Congress, 2d Session. H. of R. Ex. Doc. No. 58. Survey of the Upper Mississippi River. Letter from the Secretary of War transmitting Report of the Chief of Engineers with General Warren's Report of the Surveys of the Upper Mississippi River and its Tributaries. Washington, 18—. 8vo., 116 pp.

Annual Report upon the Improvement of the Mouth of the Mississippi; Removal of the Red River Raft; Improvement of Rivers and Harbors in the States of Louisiana and Texas, in charge of Capt. C. W. Howell, U. S. A., being Appendix Q of the Annual Report of the Chief of Engineers to the Secretary of War for 1873. Washington, 1873. 8vo., 101 pp.

Report of the Commission of Engineers upon the Reclamation of the Alluvial Basin of the Mississippi River; being Appendix O of the Annual Report of the Chief of Engineers for 1875 to the Secretary of War. Washington, 1875. 8vo., 146 pp., and 5 maps.

Annual Report upon the Improvements of Rivers and Harbors in the District of Columbia, Virginia, and North Carolina, in charge of S. T. Abert, being Appendix G of the Annual Report of the Chief of Engineers for 1876 to the Secretary of War. Washington, 1876. 8vo., 93 pp., and 4 maps.

MISSOURI.

*Swallow's Third Report also published. 1857. 6 pp.

NEW BRUNSWICK.

Gesner's Second Report contains 76 pp.

NEWFOUNDLAND.

Geological Survey of Newfoundland, Alexander Murray, Director, and James P. Howley, Assistant. Report of Progress for the year 1877. St. John's, 1878. 8vo., 13 pp.

NEW JERSEY.

In Executive Documents. Geological Survey, Report for 1857.

Reports of William Kitchell, Maurice Beesley, George H. Cook, and Egbert L. Viele. Trenton, 1858. 8vo., 10 pp.

Geological Survey of New Jersey. Annual Report of the State Geologist for the year 1879. Trenton, 1879. 8vo., 199 pp., and 2 maps. [George H. Cook.]

New York.

State of New York. No. 374. In Assembly. April 18th, 1835. Report of the Select Committee on the Memorial of the American Institute. Albany, 1843. 8vo., 7 pp. [In regard to geological survey.]

State of New York. No. 59. In Senate. March 6th, 1843. Communication from Mr. James Hall, one of the State Geologists. Albany, 1843. 8vo., 9 pp.

State of New York. No. 60. In Senate. March 7th, 1843. Communication from Messrs. Emmons and Hall, State Geologists. Albany, 1843. 8vo., 9 pp.

Catalogue of the Cabinet of Natural History of the State of New York, and of the Historical and Antiquarian Collection annexed thereto. Albany, 1853. 8vo., 34, 61, 22, 53, 31, 28, and 21 pp.

Third Regents' Report, Revised Edition, contains 183 pp.

Fourteenth Regents' Report contains 84 and 109 pp., and 20 plates.

Twenty-eighth Regents' Report. [State Museum Edition. Albany, 1879. 8vo., 210 pp., and 37 plates.]

Thirtieth Annual Report on the New York State Museum of Natural History by the Regents of the University of the State of New York. Albany, 1877. 8vo., 117 pp., and 4 plates.

[State Museum Edition. Albany, 1879. 8vo., 256 pp., and 4 plates.]

Thirty-first Annual Report on the New York State Museum of Natural History by the Regents of the University of the State of New York. Albany, 1879. 8vo., 78 pp., and plate.

*Vol. I. Memoirs of the Board of Agriculture of the State of New York. Memoir I. Survey of Albany County; by Eaton and Beck. Albany, 1821.

*Vol. II. Contains "A Geological and Agricultural Survey of Rensselaer County;" by Amos Eaton. pp. 3–43.

*Also "A Report of the Geological Structure of the County of Saratoga;" by Dr. John H. Steel. pp. 44–84, and pp. 155–161. Albany, 1823.

*Vol. X. Transactions New York State Agricultural Society.

Albany, 1851. Contains "Agricultural Survey of the County of Seneca;" by John Delafield. 8vo.

*Vol. XI. Transactions New York State Agricultural Society. Albany, 1853. Contains "Surveys of Essex County;" by Winslow C. Watson. 8vo.

*Vol. —. Transactions New York State Agricultural Society. Albany, 1860. Contains "Survey of Onondaga County;" by W. Kelley and others. 8vo.

North Carolina.

Olmstead's two reports are printed small 4to. and not 12mo.

For Rothe's and Mitchell's Reports insert:

Report on the Geology of North Carolina, conducted under the direction of the Board of Agriculture. Part III. By Elisha Mitchell. November, 1827. Small 4to., 27 pp.

Gold Mines. The following Remarks on the Gold Mines of this State, by Charles E. Rothe, Miner and Mineralogist from Saxony, are copied from Prof. Silliman's American Journal of Science. November, 1827. Small 4to., 15 pp.

XXXVth Congress, 2d Session. Senate Ex. Doc. No. 26. Report of the Secretary of the Navy, communicating the Report of Officers appointed by him to make the Examination of the Iron, Coal, and Timber of the Deep River Country, in the State of North Carolina, required by a Resolution of the Senate. Washington, 1859. 8vo., 29 pp., section, and 2 maps.

Report of the Progress of the Geological Survey of North Carolina, 1866; by W. C. Kerr. Raleigh, 1867. 8vo., 56 pp.

Swamp Lands of the State of North Carolina: Facts for Emigrants and Capitalists. Published by order of the Literary Board. Raleigh, 1867. 8vo., 31 pp.

Nova Scotia.

Hunt's Report on the Gold Region of Nova Scotia contains but 38 pp.

Ohio.

Catalogue of the Geological Specimens collected on the late Survey of the State of Ohio; by W. W. Mather, State Geologist. February 25th, 1842. 8vo., 7 pp., and 11 tables.

*Report on the State House Artesian Wells, at Columbus, Ohio; by W. W. Mather. Columbus, 1859. 8vo., 41 pp.

*Minority Report on the Expenditures of the Geological Survey of the State; by the Standing Committee on Retrenchment. Columbus, 1869. 8vo., 8 pp.

Report of Progress for 1869. [Reprinted. Columbus, 1871. 8vo., 176 pp., 4 sections, and 2 maps.]

*Six sheets of the Geological Atlas of Ohio were issued in 1879.

PENNSYLVANIA.

Second Geological Survey of Pennsylvania, 1876–78, MM. Second Report of Progress in the Laboratory of the Survey at Harrisburg; by Andrew S. McCreath. Harrisburg, 1879. 8vo., xi and 438 pp., and 3 plates.

Second Geological Survey of Pennsylvania. Report of Progress, P. Atlas to the Coal Flora of Pennsylvania, and of the Carboniferous Formation throughout the United States; by Leo Lesquereux. Harrisburg, 1879. 8vo., 18 pp., and 87 plates.

Second Geological Survey of Pennsylvania. Report of Progress, PP. The Peruvian or Upper Carboniferous Flora of West Virginia and Southwestern Pennsylvania; by William M. Fontaine and I. C. White. Harrisburg, 1880. 8vo., ix and 143 pp., and 38 plates.

Second Geological Survey of Pennsylvania. Report of Progress in 1877, QQ. The Geology of Lawrence County. To which is appended a Special Report on the Correlation of the Coal Measures in Western Pennsylvania and Eastern Ohio; by I. C. White. Harrisburg, 1879. 8vo., xxxvi and 336 pp., 2 plates, and map.

Second Geological Survey of Pennsylvania. Report of Progress in 1878, QQQ. The Geology of Mercer County; by I. C. White. Harrisburg, 1880. 8vo., xiv and 233 pp., and map.

Second Geological Survey of Pennsylvania. Report of Progress, V. Part First: The Northern Townships of Butler County. Part Second: A Special Survey made in 1875 along the Beaver and Shenango Rivers, in Beaver, Lawrence, and Mercer Counties; by H. Martyn Chance. Harrisburg, 1879. 8vo., xvii and 248 pp., plate, and 6 maps.

RHODE ISLAND.

*A Report of the Important Hearing on the Memorial of the New

England Coal Mining Company for encouragement from the State; and on the Numerous Petitions of the Freeholders in aid of the same; before the Select Special Committee of the General Assembly of Rhode Island and Providence Plantations. Together with the Report of the Committee unanimously adopted by the Assembly in favor of the Prayer of the Memorialists and Petitioners, and of a Geological and Agricultural Survey of the State in 1838. New England, 1838. 8vo., viii and 148 pp.

Rocky Mountain Region.

Lewis and Clark's Report, for "folding title" insert "folding table."

Nicollet's Report. Washington, 1843. Insert "and map."

Fremont's Report. Washington, 1843. Insert "8vo., 207 pp., map, and 6 plates."

Erase *Explorations in the Dakota Country in the year 1853; by Lieut. G. K. Warren. Washington, 1855. 8vo., 79 pp., and maps.

In Warren's Report on Dakota Country, Washington, 1856, erase "plates."

In Warren's Preliminary Report of Explorations in Nebraska and Dakota, for "Washington, 1858," read "Washington, 1859."

Erase XXXIVth Congress. H. of R. Doc. No. 129. Washington, 1855, 43 pp.

In Vol. III, Pacific Railroad Reports, insert i after x.

In Vol. X, Pacific Railroad Reports, change 20 to 24 pp.

In Volume XI, Pacific Railroad Reports, change 120 to 115 pp.

Bryan's Report is in H. of R. Ex. Doc. No. 2, and not No. 11.

Mullan's Wagon Road Report, Walla-Walla to Benton, contains 8 instead of 7 plates.

Department of the Interior. Report of the U. S. Geological Survey of the Territories. Vol. XI. Monographs of North American Rodentia contains 7 instead of 6 plates.

Report upon the Reconnoissance of the Yellowstone National Park; made in 1873 by Capt. William A. Jones. A portion of this report was issued in advance as House Doc. No. 285. Washington, 1874. Also with Ludlow's 1874, Report. 8vo., 124 pp., and 3 maps, as Appendix PP. to Report of the Chief of Engineers to the Secretary of War for 1875.

*XXVIIIth Congress. 1st Session. Senate Ex. Doc. No. 401. Report of the Secretary of War in Relation to the Survey of the Strait connecting Lakes Huron and Erie, and the Improvement

of a Channel in the Straits of Detroit. June 14th, 1844. Washington, 1844. 8vo., 3 pp., and map.

XXIXth Congress. 1st Session. Senate Ex. Doc. No. 438. Message from the President of the United States communicating a Report of an Expedition led by Lieut. Abert on the Upper Arkansas and through the Country of the Comanche Indians, in the fall of the year 1845. Washington, 1845. 8vo., 75 pp., 12 plates and map.

*Report of Expedition from Fort Leavenworth to Fort Vancouver; by Osborne Cross. May 20th to October 5th, 1850. Washington, ——. 8vo., 127 to 244 pp., and 36 plates.

*Journal of an Expedition to the Mauvais Terres and the Upper Missouri, in 1850; by Thaddeus A. Culbertson. Washington, 185–. 8vo., pp. 84 to 145.

*House Document No. 129. Explorations and Surveys for a Railroad from the Mississippi River to the Pacific Ocean. War Department. Appendix to the Preliminary Geological Report of William P. Blake. Palæontology. Washington, 1853. 8vo., 34 pp.

Report of the Secretary of War for 1876 contains Report of Capt. A. A. Humphreys, U. S. A., "Upon the Progress of the Annual Report upon Explorations and Surveys in the Department of the Missouri; by Lieut. E. H. Ruffner, U. S. A.;" being Appendix QQ of the Annual Report of the Chief of Engineers to the Secretary of War for 1876. Washington, 1876. 8vo., 35 pp., and map.

[In this is included XLIVth Congress. 1st Session. H. of R. Ex. Doc. No. 172. Report on Lines of Communication between Southern Colorado and Northern New Mexico.]

Annual Report upon Explorations and Surveys in the Department of Dakota; by Lieut. Edward Maguire, U. S. A.; being Appendix PP of the Annual Report of the Chief of Engineers to the Secretary of War for 1877. Washington, 1877. 8vo., pp. 1337 to 1380, and 10 plates.

Annual Report upon Explorations and Surveys in the Department of Dakota; by Lieut. Edward Maguire, U. S. A.; being Appendix QQ of the Annual Report of the Chief of Engineers to the Secretary of War for 1878. Washington, 1878. 8vo., pp. 1671 to 1703.

Wheeler's Annual Report for 1878 contains x and 234 pp., and 7 plates, with Atlas of 9 maps., viz.; Topographical atlas sheet

84 B, and Land Classification Series 41B, 61C, 62A, 62C, 69D, and 77D.

*Hayden's U. S. Geological Survey of the Territories published 5 maps in 1879.

XLVth Congress. 2d Session. H. of R. Ex. Doc. No. 80. Geological and Geographical Surveys. Letter from the Secretary of the Interior transmitting Report of Professor Powell in regard to Surveys, in response to a Resolution of the House of Representatives. Washington, 1878. 8vo., 19 pp., and map.

*Pacific Railroad Explorations and Surveys. Washington, 1856. 8vo., pp. 203 to 216.

Annual Report of Capt. A. A. Humphreys, Topographical Engineer, in charge of Office Explorations and Surveys, War Department, December, 1858. Washington, 1859. 8vo., 173 pp., and map. [Contains Warren's Report, just above; Lieut. J. C. Ives's Preliminary Report of Colorado Exploring Expedition, on next page but one, and Capt. John Pope's Report on Artesian Wells Experiments, next page but one.]

*Preliminary Report of the General Features of the Military Reconnoissance through Southern Nevada, conducted under directions of Lieut. George M. Wheeler, assisted by Lieut. D. W. Lockwood, 1869. Washington, 18—. Royal 8vo., 25 pp., and map.

Department of the Interior. U. S. Geographical and Geological Survey of the Rocky Mountain Region. J. W. Powell in charge. Report of the Geology of the Henry Mountains; by G. K. Gilbert. Washington, 1877. 4to., x and 160 pp., 22 plates, and 5 maps.

XLVth Congress. 2d Session. H. of R. Ex. Doc. No. 73. The title should be: XLVth Congress. 2d Session. H. of R. Ex. Doc. No. 73. Report on the Lands of the Arid Region of the United States, with a more detailed account of the Lands of Utah; by J. W. Powell. Washington, 1878. 4to., xiii and 195 pp., and 3 maps.

Department of the Interior. Bulletin of the United States Geological Survey of the Territories. F. V. Hayden, U. S. Geologist, in charge. Vol. V, No. 2, Washington, 1879. 8vo., 178 pp., and 4 plates. Vol. V, No. 3, Washington, 1879. 8vo., 190 pp.

XLVth Congress. 2d Session. H. of R. Ex. Doc. No. 88. Surveys by the War Department. Letter from the Secretary of War in response to a Resolution of the House of Representatives,

giving information concerning the Surveys conducted by the Department during the last ten years. Washington, 1878. 8vo., 8 pp. and maps.

Views of the War Department concerning the Public Surveys of the Territories of the United States; being Appendix NN of the Annual Report of the Chief of Engineers to the Secretary of War for 1878. Washington, 1878. 8vo., pp. 1661 to 1666.

XLVth Congress. 3d Session. H. of R. Ex. Doc. No. 62. Organization of Coast and Geodetic Survey. Letter from the Secretary of the Treasury in response to a Resolution of the House of Representatives concerning the Present Organization of the Coast and Geodetic Survey. Washington, 1879. 8vo., 6 pp.

XLVth Congress. 3d Session. H. of R. Ex. Doc., No. 104. Surveys west of the One Hundredth Meridian. Letter from the Secretary of War transmitting a statement in regard to the Total Amount Expended in the Prosecution of Surveys west of the One Hundredth Meridian. Washington, 1879. 8vo., 2 pp.

XLVIth Congress. 1st Session. Senate Ex. Doc. No. 11. Letter from the Secretary of War transmitting a Report of the Acting Chief of Engineers upon a Provision in the Sundry Civil Bill, approved March 3d, 1879, discontinuing the Geographical Surveys west of the One Hundredth Meridian, under the War Department, after June 30th, 1879. Washington, 1879. 8vo., 3 pp.

TENNESSEE.

The State Papers of Tennessee for 1859 contain: Third Biennial Report on the Geological Survey, by J. M. Safford. Nashville, 1859. 8vo., pp. 297 to 302.

Statement made by the State Geologist to the Thirty-fourth General Assembly of Tennessee, adjourned Session, 1866, with reference to the condition of the Geological Report ordered to be printed, February 7th, 1860. Nashville, 1866. 8vo., 7 pp. [J. M. Safford.]

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VERMONT.

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VIRGINIA.

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WISCONSIN.

Annual Report of the Wisconsin Geological Survey for the year 1879; by T. C. Chamberlin, Chief Geologist. Madison, 1880. 8vo., 72 pp.

The State of Wisconsin, etc. Madison, 1876. Contains a map.

Geology of Wisconsin. Survey of 1873 to 1879. Vol. III. Madison, 1880. Royal 8vo., xxxii and 763 pp., 53 plates, and an atlas of 14 maps. [R. D. Irving, R. Pumpelly, C. E. Wright, E. T. Sweet, M. Strong, T. C. Chamberlin, A. Wichmann, and T. B. Brooks.]

THE MINERAL RESOURCES OF WISCONSIN.

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THE object of the present paper is to give an outline account of the mineral resources of the State of Wisconsin, so far as they are now known, including both metallic ores and non-metallic useful mineral substances. The only previous publication making any attempt at such an account is one of a popular nature, prepared by the writer for Snyder and Van Vechten's *Historical Atlas of Wisconsin*, which was published at Milwaukee in 1878. The data upon which the statements of the following pages are based are chiefly from the

reports of the recently completed Wisconsin Survey, including Professor T. C. Chamberlin's report on Eastern Wisconsin, the late Moses Strong's reports on the Lead Region and on the Upper St. Croix District, the writer's reports on Central Wisconsin and on the Eastern Lake Superior District, E. T. Sweet's report on the Western Lake Superior District, and the reports of T. B. Brooks and C. E. Wright on the Menominee Iron Region. The mode of presentation of facts, conclusions, and opinions are of course original. Since the larger number of the members of the Institute are not likely to possess copies of the Wisconsin reports, it is thought that the account will be of use to them; while even for those who do receive the reports, since these contain no such abbreviated summary, it is thought that the paper will have some value.

SKETCH OF THE GEOLOGICAL STRUCTURE OF WISCONSIN.

The region of country embraced within the boundaries of the State of Wisconsin may be described as consisting of a nucleus of very ancient crystalline schists, surrounded on three sides by beds of undisturbed and unaltered limestone and sandstone belonging to the Silurian system; the fourth or Lake Superior side being bounded by a broad belt of a system of interbedded volcanic and fragmental rocks, wholly unknown elsewhere.* The Silurian formations on the east, south, and west dip gently away from the nucleus in all directions, coming to the surface in a series of roughly concentric bands. A brief description is given below of each of the several rock formations of the State, the accompanying map† and section serving suffi-

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* See footnote, p. 486.

† The topography, political geography, railroads, etc., of this map are from the large wall map of Wisconsin by Professors Nicodemus and Conover, the only one yet published of this State having any value. Portions of Minnesota, Michigan, and Iowa have been added from other sources; so far as possible from the geological maps of those States. The geological formations for Wisconsin are chiefly from the atlas plates of the Wisconsin Survey, including general "area" maps (three miles to the inch), by T. C. Chamberlin, R. D. Irving, M. Strong, E. T. Sweet, and L. C. Wooster; and special maps of the Lead Region (one mile to the inch), by M. Strong, of the Eastern Lake Superior District (two miles to the inch), the Penokee Iron Range (3.6 inches to the mile), and the Penokee Fault ($\frac{1}{3520}$), by R. D. Irving, and of the Menominee Iron Region ($\frac{3}{4}$ inch, .375 inch, and $\frac{1}{3}$ inch to the mile), by T. B. Brooks and C. E. Wright. These maps leave untouched a large area in the heart of the northern part of the State. Most of this region is known to be underlaid by Laurentian gneiss, granite, and schists; but near its borders are areas of Huronian, whose inner boundaries are not always

ciently to show the surface distribution and structural relations of each.

Laurentian System.—The rocks of the greater part of the crystalline nucleus are referred to the Laurentian system of the Canadian geologists, on the ground of close lithological similarity—the only marked difference being the absence of crystalline limestones in the

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definitely known. The large Huronian area on the west, including a small number of quite distant outcrops, and in a region largely covered by drift, is copied as to its extent and boundaries from a small map published by Professor T. C. Chamberlin in Snyder and Van Vechten's Atlas of 1878. On the east side of the Laurentian core Wright has left the inner boundary of the Huronian indefinite. From the facts given by him on atlas plate XXX of the Wisconsin Report, from a published account of a trip on the Wisconsin and Wolf rivers by Sweet (Trans. Wis. Acad. of Sci., vol. iii), and from my own personal knowledge of the schistose rocks of the Valley of the Wisconsin in Marathon County, I have mapped this boundary as given. For the portion of Michigan included within the map the geology is taken from the atlas of the Michigan Survey, with two changes. These are the inclusion of a large area of sandstone on the northern side of Keweenaw Point, marked doubtful in the atlas, within the Keweenawan System; and the extension of the Huronian and Laurentian over an area near the Wisconsin line left uncolored in the atlas. The latter change, or rather addition, is warranted by the more recent investigations by Brooks (see atlas plate XXIX, Wisconsin Report), by Foster and Whitney's map, and by the strong evidence of direct continuity—as afforded by their close lithological and stratigraphical similarity—of the Marquette and Penokee Huronian areas. For Minnesota, the geology of Mower, Fillmore, Houston, Dodge, Olmstead, and Ramsey counties is from N. H. Winchell's recent maps. For the remainder of that portion of Minnesota bordering the Mississippi I have used the large map of the British Provinces and the Northern United States published by the Canada Survey in 1866. For Northern Minnesota the geology has been sketched in from a comparison of the results of the Wisconsin Survey in adjacent territory, of the Canadian map just referred to, and of the map and reports of D. D. Owen and his associates. The extension of the Keweenawan system into Minnesota, south of the St. Louis, is a thing not recognized heretofore by the Minnesota geologists, and it is only possible to map it roughly. Still the structural relations of the system, and the facts given in the reports of D. D. Owen, the members of whose corps ascended or descended nearly all the streams of this region, show that our mapping cannot be far wrong. The northern limits of the Huronian in Minnesota are left indefinite. For Iowa, I have used J. D. Whitney's map of the Upper Mississippi Lead Region, the large map of the Canada Survey, and the maps of the Iowa Survey, the latter being so exceedingly small and general as to be of little assistance. For Illinois I have used Whitney's map and the large wall map of the Illinois Survey, issued in 1875. It was not always easy to make the more accurate Wisconsin maps fit with that of Illinois, which is a disgrace to that State, both in appearance and in the loose and general style of mapping it indicates.

Chamberlin has in preparation for the first plate of the atlas of the Wisconsin Survey, a geological map of Wisconsin, on a much larger scale and in

Wisconsin area*—of similar structural relations to the Huronian, Keweenawan, and Lower Silurian systems, and of probable direct continuity with the Canada Laurentian through the upper peninsula of Michigan and underneath the waters of Lake Superior. The Wisconsin Laurentian lies in a very irregularly outlined area, whose extreme dimensions from north to south are about 135 miles. The northern border of this area is only 12 miles from Lake Superior at the nearest point of approach, the southern edge is about 140 miles from the south line of the State, the eastern 25 to 30 miles from Lake Michigan, and the western some 70 miles from the St. Croix River, the western boundary of the State. A small portion of this area, on its northeastern edge, extends over the State line into the upper peninsula of Michigan, but it is completely encircled by newer formations, being bordered the greater part of its circumference by the Potsdam sandstone. For a long distance on the north and northeast, and again on the west, it is limited by the Huronian schists, and for a shorter distance on the northwest, by the Keweenawan rocks. As shown subsequently, however, there is a complete border, in part of great width, of Huronian, though it is in places overlapped by the newer formations.

The highest portion of the Laurentian area lies near its northern edge, where an elevation is reached of from 900 to 1200 feet above Lake Superior, and of 1500 to 1800 feet above the sea. This is also the highest land in the State, excepting a few isolated peaks and the Penokee ridge of the Huronian, just north, the latter having about the altitude mentioned. From this high northern portion there is a very rapid slope to Lake Superior. Southeast towards Lake Michigan, and southwest towards the Mississippi River depressions, the slopes are much more gradual, reaching on the edge of the Laurentian area an altitude of about 900 feet above the sea. Between these last-named slopes there is a well-defined north and south dividing ridge, which itself slopes southward, and is recognizable to the south line of the State, the Silurian formations arching over it.

greater detail than the one presented herewith, which, however, is peculiar in the inclusion of portions of the adjoining States, and in the indications of the mining districts.

* Mr. Selwyn now regards the formation of Canada in which the great limestone bands occur as Huronian, and restricts the term Laurentian to "all those clearly lower unconformable granitoid gneisses, in which we never find interstratified bands of calcareous, argillaceous, arenaceous and conglomeratic rocks," which description would apply exactly to our Wisconsin Laurentian. (See Report of the Canada Geological Survey for 1877–78.)

In general the Laurentian district has only a very gently undulating surface, which is, however, at times broken in minor detail by low ridges, with outcropping tilted rock ledges or rounded bosses of bare granite. A few points of harder rocks stand up boldly above the general level. Taking the region as a whole, rock exposures are not frequent, being most commonly found in the beds of the streams, many of which have their courses frequently broken by rock rapids. Drift materials are abundant over most of the region, commonly concealing the rocks. In the northern and northeastern portions they reach an enormous development, causing the surface to be everywhere dotted with small lakes without outlet, the curious depressions in which these almost innumerable lakes lie resulting from the irregularities incident upon the morainic method of deposition. Nearly the whole region is covered with a dense forest, chiefly of pine, though occasionally with hard wood ridges interspersed. It is one of the principal lumbering districts of the United States, but away from the several flourishing towns supported by the lumber interest it is in the main a complete wilderness.

The most common rock of the Laurentian nucleus is gneiss, including under this general name, however, many varieties. Along the northern border of the area, a dark-colored hornblende-gneiss, often highly chloritic from a change of the amphibole, and a pinkish, quartzose, gneissoid granite are the prevailing rocks In the more southern portions mica-gneisses of many varieties are much more common. Hornblende-schists and hornblende-rocks of several kinds are abundant. Diorite occurs, and intrusive diabase, often indistinguishable macroscopically from the diorite, is somewhat less common. True granites are very abundant, occurring both as a phase of and grading into the gneiss, and in large, independent, intrusive masses. The granites vary greatly in character, running from very fine to very coarse-grained, and include both hornblendic and micaceous varieties, as well as kinds in which both or neither hornblende and mica are present. A few other kinds of rocks are known, but are rarely met with. Quartzites and quartz-schists occur at several points within what has been called the Laurentian, but in nearly every case their structural relations are such as to render it probable that they are Huronian, and all known quartzite areas have accordingly been included with the Huronian on the accompanying map. The whole system is distinctly bedded, and pressed into a series of close folds. The intricate folding, probable numerous dislocations, common drift covering, and wilderness-like nature of the region have thus far prevented

any attempt being made towards unravelling the subordinate structure, except along the lines of a few of the main streams.

It is somewhat curious that no ores have ever been discovered within the Laurentian district, since similar rocks in other regions are quite commonly associated with metallic ores. In Canada the same system includes extensive iron deposits and profitable gold veins, as well as a number of other useful mineral materials. The writer has detected minute quantities of gold and silver in quartz from Clark County, and has tested a considerable number of favorable and unfavorable looking quartz samples, but with negative results. Though there have not as yet been found any ores, there is no inherent improbability of their discovery in the future. As yet ornamental granites and kaolin are the only substances of economic value known to exist in connection with these rocks.

Huronian System.—Into the northeastern part of Wisconsin, in the vicinity of the Menominee River, extends the well-known schistose iron-bearing series of the upper peninsula of Michigan, with its characters well preserved. The rocks of this series have been called Huronian by Brooks, and, in the writer's judgment, correctly so, on account of their similarity to the Canada Huronian, with which they not improbably have a direct connection underneath the Silurian of the eastern part of the peninsula, but more especially because they evidently occupy the same geological interval as the typical Canadian series, exhibiting the same non-conformity with an underlying gneissic and granitic system. The absurdity of the attempt made by some to refer both of the systems here called Laurentian and Huronian to the Silurian becomes plain when the fact is understood that both, as well as a newer series, are overlaid unconformably by unaltered and undisturbed sandstones holding Primordial fossils, and often full of pebbles and fragments broken from the older rocks. The rocks of the Huronian of the Menominee region are, with the single exception of granite, highly schistose, and include more especially hornblendic and mica schists, clay slates, chloritic schists, actinolite schists, gneiss, limestones, diorite, diabase and iron ores. These rocks are very intricately folded. They have been studied in detail by Brooks, who thinks that he recognizes in them the same set of beds that he had made out previously with so much skill and labor in the Marquette region, though in such a complicated and often drift-covered district, that the mapping of the folds has to be in large measure hypothetical, even after all ascertainable facts are in. The inner boun-

dary of this Huronian area has been left undetermined by Brooks and Wright. The formation evidently, however, has a wide extent westward beyond the region studied by them, and I have so represented it, as explained in a footnote on a previous page.

On the north side of the Laurentian nucleus is again a belt of schistose rocks, stretching westward from Lake Gogebic, in Michigan, for over eighty miles, to Lake Numakagon, in Wisconsin. These, in lithological and general stratigraphical characters, approach so closely to Brooks's description* of the Marquette Huronian, while at the same time demonstrably non-conformable with the gneissic Laurentian just south, that there can be no doubt that they belong to the same system. The striking similarity in stratigraphical arrangement seems to demonstrate the actual continuity of the two systems, a thing not yet proved by actual observation. In the Penokee region the rocks all dip north,† for the most part at a high angle (50° to 70°), though this lessens towards the west, and the subordinate beds trend with the general course of the belt. In a condensed form the succession in this region of the Huronian beds—some of which have been traced for fifty miles—may be stated as follows, beginning with the lowest or oldest belts: (1) crystalline tremolitic limestone, with a subordinate bed of quartzite, 130 feet; (2) straw-colored to greenish siliceous schists, often novaculite, 410 feet; (3) tremolitic or actinolitic magnetite-schists, lean magnetic and specular iron ores, magnetitic and specular quartzites, forming the bulk of the Penokee iron range, 780 feet; (4) alternations of black mica-slates, diorites, schistose quartzites, and as yet unfilled gaps, 3495 feet; (5) medium grained to aphanitic, light to dark-gray mica (biotite) schists, with coarse, intrusive biotite granite, 7985 feet; in all 12,800 feet. Westward from Bad River (Sec. 14, T. 44, R. 3 W.) the gabbro forming the base of the Keweenawan system gradually cuts across the upper portion of the Huronian, narrowing the Huronian belt more and more, until, west of Lake Numakagon, the Keweenawan diabase and Laurentian granite are in contact, and the Huronian belt has disappeared altogether. This disappearance is supposed to

* Geological Survey of Michigan, vol. i, 1873.

† Brooks (vol. iii, Geology of Wisconsin, p. 446, footnote) expresses some surprise that I have worked out no folds in the rocks of the Penokee region. Very brief study of my report on this region will convince him that they do not exist; and a simple glance at a map showing how very narrow is the belt occupied by the Penokee Huronian, and its relation to the Lake Superior trough, will prove to him that they could not exist.

be due to the covering of the Huronian by the Keweenawan volcanic rocks, it being probable that the former system continues underneath far to the westward, joining finally, in Minnesota, with the schistose rocks of the St. Louis and of the north shore of Lake Superior. The St. Louis schistose rocks, which are exposed on a large scale above and below the railroad crossing at Thompson, dip southward, and are to be regarded as forming the north side of a grand synclinal, in which many minor folds may be present, and of which the Penokee system is the southern side.

On the west side of the Laurentian nucleus, in Barron County, —T. 34, R. 11 W., and T. 35, R. 10 W.,—as long since shown by Owen, are again quartzites, associated with the peculiar red aluminous rock called pipestone, from which the Indians made their well-known pipes, and which is known to occur in one other place only in the United States, viz., in Southwestern Minnesota, near the Sioux River. South and east from here, in T. 32, R. 6 W., is a bold ridge of quartzite and quartzite conglomerate, with a thickness in all of some 5000 feet, and apparently unconformably placed on the Laurentian gneiss, which shows in the immediate vicinity. Professor Chamberlin maps this quartzite as part of the same Huronian area with the quartzites of Barron County, and in this I have copied from him on the accompanying map.

South and east from the last-named place, on Black River, T. 21, R. 4 W., most of the intervening space showing the overlying Potsdam sandstones as the surface rock, the Huronian schists appear again. They are seen here along the bed of the stream, which has worn its way down to them through the cover of sandstone, through which also they project in a few points away from the river. The rocks here are quartz-schists, often very highly charged with magnetite or specular iron, unctuous (magnesian?) schists and granite.

Continuing our circuit around the Laurentian nucleus, we find the rest of its southern edge bordered by the Potsdam sandstone, through which, however, the Huronian rocks rise in a number of places in projections of greater or less size. These projecting points indicate a wide spread southward of the Huronian, the southernmost point of their occurrence being as far south as T. 7, R. 13 E., in Jefferson County. Their arrangement, moreover, as will be readily perceived from the accompanying map, indicates a return northward of the Huronian belt to join with the schists of the Menominee region, thus producing a complete band.

Of these isolated Huronian areas, the most noteworthy are the

Baraboo ranges of Sauk and Columbia counties. These are two bold ridges, united at their extremities, having a length of some five and twenty miles, and composed chiefly of quartzite, with which, however, there is associated a great thickness of quartz-porphyry. The latter rock constitutes the whole of a number of the other isolated areas, a fact of much interest, since in this respect the more southern Huronian of Wisconsin approaches closely the rocks which in Missouri hold the famous specular iron deposits of that region, while the Huronian of the northern part of the State is close in all its characters to the formation that holds the equally famous ores of the Marquette region of Michigan.

The principal mineral substances of economic value contained in the Huronian of Wisconsin are iron ores.

Keweenawan System.—This remarkable series of rocks, peculiar to the Lake Superior region,* has in Wisconsin alone an area of not far from 10,000 square miles. In the region of Keweenaw Point, and also on the northern side of the lake, it has long attracted much attention from the extraordinary deposits of copper contained in it. Its extent in Wisconsin has, however, been only vaguely known, and it has been left to the recent Survey to map it out exactly, and to demonstrate a far greater extent than hitherto suspected, the chief credit for this addition to our knowledge belonging to the late Moses Strong.

Beginning on Keweenaw Point we find a belt of these rocks, made up of interbedded fragmental (sandstone and conglomerate) and volcanic (all plagioclase-augite) rocks, extending all along the lake shore to the Wisconsin line. The detrital beds predominate to the north and the fragmental to the south of this belt, and all dip con-

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* Unless, indeed, Mr. Selwyn's reference of part of the Quebec group of Logan to this system is correct (Geological Report of Canada for 1877–78). He certainly makes out a strong case in his complete rearrangement of this previously heterogeneous and problematical group. I cannot, however, agree with him as to the uselessness of the word "Keweenawan." This system has characters so sharply contrasting with those of the Huronian, besides being, as it now seems to me, unconformable with it, as developed in the basin of Lake Superior, that it is fairly entitled to a separate name. The one fact of the common alteration of the rocks of the Huronian or schistose series, and the entire absence of any such metamorphism in the fragmental beds of the Keweenawan, is enough to separate them from each other.

It should also be stated that Bell's recent investigations on the east coast of Hudson's Bay have brought to light a great system of interbedded volcanic and fragmental rocks, occupying the stratigraphical position of the Keweenawan, with which they have also many lithological characteristics in common.

stantly northward, the angle varying, but reaching verticality on the Wisconsin line. Along its whole course this belt is characterized by the occurrence of native copper, either in veins, or as impregnations of the greatly altered amygdaloidal diabases, or as impregnations of the porphyry conglomerates. The counterpart of this belt, with its members dipping southward, and the relative positions of those portions in which the fragmental and volcanic members respectively predominate reversed, is found in the rocks of Isle Royale, as long ago pointed out by Foster and Whitney. The same system of rocks is also largely developed on the northwestern shores of the lake, for the most part with an inclination towards the lake basin. Following the Keweenaw Point belt into Wisconsin we find it leaving the lake shore and passing far inland, stretching all across the State of Wisconsin and into Minnesota, with the dip steadily lessening, however, until in the vicinity of the St. Croix the great volcanic flows are nearly horizontal and spread over a much wider area than anywhere else throughout the whole extent of the formation.

A second prominent ridge, composed of Keweenawan rocks in all respects similar to those of the Keweenaw Point belt, forms the backbone of the Bayfield peninsula, and stretches thence westward into Minnesota. The rocks of this belt dip *southward*, and form the northern side of a synclinal of which the Keweenaw Point belt forms the southern side. This synclinal is evidently the same as that lying between Isle Royale and Keweenaw Point, which thus, in its western extension, passes entirely on to the land, the western end of the lake basin being carved altogether in southward dipping beds.

Chaquamegon Bay lies within the synclinal, which, for a short distance inland, has for the surface rock the horizontal red Potsdam sandstone so well known along the south shore of Lake Superior. Further west, however, the Keweenawan rocks are seemingly without other cover than the abundant drift materials, from side to side of the synclinal. Towards the Minnesota line, where the dip on both sides of the synclinal is so much flattened, there is consequently an immense surface spread, the width of the belt of country underlaid directly by this system being here over fifty miles. Across the line in Minnesota the Keweenawan synclinal evidently soon ends in a shape like that of the end of a broad flattened spoon.

The age of this great system of rocks has been the subject of much discussion, it having long been held that it formed a part of the Potsdam sandstone series greatly disturbed and thickened by the associated volcanic rocks. Brooks and Pumpelly first gave

good reasons for a belief in the truth that had previously only been suspected, viz., that these rocks form a distinct system, placed unconformably beneath the horizontal Lake Superior or Potsdam sandstone. This belief has been carried to demonstration, as it appears to me, by the results of the Wisconsin Survey.* Beyond question this system occupies part of the great gap everywhere found at the meeting of the lowest fossiliferous rocks and the ancient crystalline schists.

The only reason that ever existed for placing the Keweenawan and Lake Superior Potsdam in the same category was the presence in the former series of sandstone resembling the rock which makes up the latter formation; and some very bold theories had to be resorted to to explain the structural relations of the two sandstones. But now that it is evident that we have two separate systems to deal with, even the lithological resemblance between the two kinds of sandstone—one of which occurs in a thickness of thousands of feet, while the other probably never reaches over 500—is seen to be more apparent than real.

The principal substance of economic value in the Keweenawan is copper, always in a native state. The belt of rocks which has been worked near Ontonagon, Michigan, for silver, belongs to this system, and extends westward into Wisconsin.

The Potsdam Sandstone Series, forming the base of the Paleozoic pile of strata, and the earliest of the fossiliferous rocks of the interior of the continent, is the great sandstone formation to which the name given originally to the lowest sandstone of New York has been transferred. The sandstone rests upon the very irregular surface of the older rocks, and therefore varies in thickness, but may be placed at about 1000 feet. The body of the formation is made up of much rolled quartz grains, often incoherent, but rather more commonly cemented by an exceedingly minute quantity of hydrous iron oxide. Towards the upper part of the formation calcareous and dolomitic particles begin to intermingle with the sand. In some regions these ingredients extend down as much as 150 to 200 feet. They aggregate, as the upper surface is approached, into more or less continuous bands, one of which, called locally the Mendota, has a very wide distribution in Central Wisconsin. Beds of greensand (glauconite) are very characteristic of the upper parts of the formation. There are some indications over a large area in Central Wisconsin of

* Geology of Wisconsin, vol. iii, pp. 23, 24, 207, 395, 417, 418.

the existence within the Potsdam of two distinct sandstone formations, the one slightly discordant with the other. Fossils are quite abundant in this formation. Most characteristic are the small phosphatic shells of *Lingulepis* and *Lingulella,* and fragments of trilobites. One trilobite of extraordinary size, the *Dicellocephalus Minnesotensis,* is especially characteristic of the Mendota beds. The horizontal sandstone of the south shore of Lake Superior belongs unquestionably to this formation, though it is a matter of doubt whether the two sandstones do or ever did connect. The Lake Superior rock differs from its more southern equivalent in its red color, which is due to large content of iron oxide, in its clayey cement (altered feldspar), and its feldspathic granules mingled with the quartz.

The economic contents of the formation are brown and hematite iron ores and building-stones.

Lower Magnesian Limestone.—This formation presents itself as a more or less massive, gray to buff-colored, irregularly stratified and cherty, dolomitic limestone, exceedingly barren in fossils. It varies in thickness from over 200 to only 50 feet, the variation being due to the fact that its upper surface was subjected to wear previous to the deposition upon it of the overlying sandstone. It appears to correspond exactly in stratigraphical position with the "Calciferous Sandrock" of the Eastern States. Its only economic value so far as known consists in its building-stone, and stone for lime-burning. Small crevices containing galena have been opened at several points in this formation, but have never proved of value, yielding only a few hundred pounds in the most favorable places.

St. Peter's Sandstone.—This formation, most commonly an incoherent quartz sandstone, is extraordinary for its very wide distribution, being known at points 250 miles apart in Wisconsin alone, while it probably is, or once was, continuous over an area whose diameters are 500 and 400 miles. In thickness it runs from a mere film to over 200 feet, having in the western and southwestern portions of Wisconsin a pretty constant thickness of 80 to 100 feet. Its only economic content is the sand of which it is composed, which is, however, frequently of so great purity and incoherence as to be well adapted for the manufacture of glass, to which use it is applied at various points in Wisconsin and Minnesota. Being very thoroughly pervious to water, being placed between two more or less completely impervious limestone formations, and lying, moreover, in the most favorable sort of position, it is, in Eastern Wisconsin, the source from which a

number of artesian wells derive their flow. It is practically non-fossiliferous.

Trenton Group.—The Trenton group is represented in Wisconsin by three formations, with a total thickness of about 500 feet. The limestones known particularly as the Trenton make up the two lower of the three subdivisions. They are yellowish to bluish, highly fossiliferous, usually thin and regularly bedded, including magnesian and non-magnesian beds, and are in all about 100 to 120 feet in thickness. A great number of fossils occur, among the most common of which are certain gasteropods and small cyathophylloid corals. The Trenton and Galena limestones together make up the lead and zinc horizon, much the greater part of the zinc ores being raised, however, from the Trenton division. Good building-stones come from this formation.

The Galena limestone is a gray or buff rock, contrasting with the Trenton below in its heavy and irregular bedding, frequent concretionary structure, and abundant chert, resembling more nearly in all these respects the Lower Magnesian. The formation includes many fossils, the most characteristic and common of which is the *Receptaculites Oweni,* often known as the "lead coral." The Galena makes up the greater part of the lead horizon. The rock is much used for making quicklime, and, to some extent, supplies a building-stone, though it is inferior in this respect to that obtained from the lower and higher limestone formations of the region.

Cincinnati Group.—This group includes a series of shales and subordinate limestones, occupying in part the horizon, and containing the fossils, of the limestone of Cincinnati. The thickness is about 200 feet. The fossils are very plenty, and include at certain points a remarkable abundance of chætetoid and bryozoan corals. Large varieties of *Orthis* and *Strophomena* are also common. The only ingredient of economic value is the clay of which the formation is itself in part made, and which in places occurs of sufficient purity to allow of its being utilized in the manufacture of brick.

Clinton Group.—The Clinton group is represented at a few points by a seam of iron ore separating the Cincinnati shales and overlying Niagara limestone. The ore constitutes one of the most important mineral resources of the State.

Niagara Limestone.—This formation underlies a very large area in the vicinity of Lake Michigan, besides occurring in a few isolated outliers in the southwestern part of the State. It includes a series of limestone beds, in all from 500 to 700 feet in thickness, and for

the most part composed of rough-textured, heavy-bedded, pure dolomite. The fossils are often very abundant, the corals and crinoids being remarkable for abundance and perfection of preservation. Its principal economic value comes from the building-stone and the very pure stone for lime-burning supplied by it. The manufacture of lime from this formation is carried on on a very large scale.

Hamilton Limestone.—This formation occupies a very small area in the immediate vicinity of Milwaukee. It is a bluish-gray to ash-colored, dolomitic, clayey limestone, only occasionally exposed, but has a high value from its hydraulic properties.

Glacial Drift.—The glacial drift occurs with an immense development in Wisconsin, though in the southwestern quarter of the State it is wholly wanting. The outline of the driftless area is indicated on the accompanying map. The boulder clay of the drift has a wide distribution, and at times, though not often, is utilized for brick-making. The pebbles of the great moraines of the eastern, central, and northern parts of the State furnish often a plentiful supply of gravel, and in numerous places are burnt for lime.

Champlain Clays.—Over large areas in the vicinity of Lakes Michigan and Superior are deposits of stratified clays, often reddish in color, and commonly highly charged with calcium carbonate. They were deposited at a time when these lakes were in a greatly expanded condition, and subsequent to the deposition of the glacial drift. These clays are very largely utilized in the making of brick, especially the peculiar white kind so much used in building in the Northwest.

IRON.

Iron mining in Wisconsin is yet in its infancy. A few very important deposits are already producing a considerable quantity of ore, and from the discoveries already made, and the now proved great extent of the Huronian or iron-bearing series of Michigan in Wisconsin, it appears probable that at no very distant day iron will be the chief mineral product of the State. We cannot predict that Wisconsin will be found to equal Michigan and Missouri in the extent and value of her iron deposits, though such a result is wholly within the possibilities. There are a number of blast furnaces in Wisconsin in the region about Green Bay, but these smelt for the most part Michigan ores.

Huronian Ores.—The Huronian of Michigan carries chiefly magnetic and specular ores, but also in some abundance the so-called soft hematites, which are usually mixtures of the red and brown

oxides, and make up a considerable proportion of the total production of that State. The Wisconsin Huronian ores, it seems probable, will be made up in about the same way.

In T. 40, R. 18 E., a short distance south of the Menominee River (the Michigan boundary), several very prominent mines have recently been opened. The most thoroughly developed and promising of these is the Commonwealth on Sec. 34. There was uncovered here in the fall of 1879, according to C. E. Wright, a total width of 104 feet in two seams of a slaty specular ore, containing upwards of 60 per cent. of iron, besides a belt of leaner ore, 10 feet wide, and containing 48 per cent.* The strata standing vertically, these figures represent nearly the true thicknesses.

Wright expresses the opinion that the shipping ore from this mine will yield from 64 to 65 per cent. The amounts of phosphorus shown by his analyses are about what are usually found in the Lake Superior ores, running from .15 to .32 per cent.,† somewhat too large a figure, according to present notions, for ores from which Bessemer pig iron is to be made. The Eagle Mine, in Sec. 20, of the same township, shows a considerable thickness of curiously associated specular soft hematite and brown ores, in alternating layers. There is a large amount of paying ore at this place, though from Wright's analysis it does not appear to be so rich as that of the Commonwealth. The ores of these two mines, and of several other openings in the same township, are regarded by Brooks as belonging to bed XV of the stratigraphical scheme which he has worked out for the Marquette and Menominee Huronian, a higher horizon than that (XIII) at which all of the rich ore of the Marquette Huronian occurs, according to the same authority.‡ Wright, however,§ places the Commonwealth ores also in XIII.

* Geology of Wisconsin, vol. iii, p. 680.

† Wright advances in his report on this region (Geology of Wisconsin, vol. iii, p. 681) the theory that the phosphorus content of all of these Lake Superior ores is derived from the percolation of surface waters, from which the iron of the ore removes the phosphoric acid, forming an iron phosphate. On this theory the iron ores would become poorer in phosphorus in depth, and to prove that this is true Wright cites a number of analytical determinations by himself on the ores from several of the Michigan mines. It will certainly take many such statistics to make geologists generally, and especially those acquainted with microscopic lithology, believe that the phosphorus of all the crystalline iron ores is not present as apatite, whose little transparent, sharply outlined crystals are known to be so commonly present in the sections of the crystalline schists in which the ore is found.

‡ Geology of Wisconsin, vol. iii, table facing p. 450.

§ Geology of Wisconsin, vol. iii, p. 679.

Several belts of magnetic attraction, at other horizons, occur on the Wisconsin side of the Menominee, and across the river in Michigan a number of successful mines are opened along a belt twenty miles in length at the horizon VI of Brooks's scheme. This belt has not yet been traced in Wisconsin. From the extraordinary developments already made in this region, from the known existence of belts of strong magnetic attraction, and from the known great extent southward and westward, in a region as yet imperfectly known, of the Huronian schists, it seems only reasonable that we should expect this part of Wisconsin to become in the future one of the most prominent iron-producing districts of the United States.

Passing now to the Huronian of the Penokee region, we find the formation extending in a narrow belt all the way from Lake Gogebic, in Michigan, to Lake Numakagon, in Wisconsin. Along the whole course of this belt the stratigraphical succession is practically the same, and the dip across the whole width, which is from half a mile to three miles, constantly to the northward. Several of the subdivisions have been traced uninterruptedly as much as fifty miles. About 540 feet above the base of the series comes in an iron-bearing belt from 800 to 900 feet wide. The rocks of this belt are chiefly magnetic quartzites and quartz-slates, the magnetite in part concentrated into narrow and very rich seams, but also often uniformly spread through the mass, and then mingled with all sorts of proportions of specular iron, up to a preponderating quantity, and often also with actinolite and tremolite. Other less common kinds occur, and towards the west end of the belt heavy diorite bands are included. Most of the belt shows much intermixed iron oxide, the magnetite and specular hematite being often mixed in a most intimate manner. A large percentage of manganese is always present, and the phosphorus and sulphur are always very low. I have sampled considerable thicknesses of this ore containing upwards of 40 per cent. of iron, and numerous narrow bands with over 60 per cent., but as yet no ore salable at the present standard of shipment in the Lake Superior region has been discovered in thicknesses great enough to be worked. Systematic exploration by trenching and test-pitting have, however, only just begun. The entire stratigraphical arrangement of the Penokee Huronian is so closely like that of the Marquette region that there can be no doubt at all that the Penokee magnetic belt, No. V of my scheme, is the equivalent of Brooks's Nos. VI to XI, which are a series of alternating diorites and magnetitic schists. If Brooks's reference of the Menominee

iron belt to VI of the Marquette scheme is correct, then that belt has undoubtedly its equivalent in the Penokee magnetic belt.

The middle portion of the Penokee Huronian is so largely drift-covered, that we are entirely ignorant as to whether the equivalent of the Marquette ore horizon is here iron-bearing or not. The rocks immediately above and below being so strikingly like those of the Marquette series, we naturally look forward to the discovery of ore here in the future, though from the negative results of the magnetic observations made, the magnetic ores are not likely to be the kinds found. It is well known that the rich specular and soft hematite ores hardly ever outcrop.

Iron-bearing rocks have recently been reported as existing on the west side of the Laurentian nucleus, in the vicinity of the Chippewa River, but nothing definite is known about them as yet. South and east of here, on Black River, the Huronian schists are highly ferruginous, carrying all of the iron oxides,—magnetic, specular, red, and brown,—but the iron content is never over 25 to 35 per cent. An iron furnace was once erected here, and an attempt made to smelt the ores. After an examination made in 1873, I came to the conclusion that there was no reasonable prospect of the discovery of workable ore here. Even if it exists it is unlikely to be discovered, since the overlying Potsdam sandstone nearly everywhere conceals the Huronian.

In the Baraboo region of Sauk County large bunches of brilliant specular iron in veins of white quartz are often met with, but no indication of ore in quantity has ever been observed. It is a matter of great interest that while we have in the Penokee and Menominee regions the same kinds and succession of rocks as in the iron district of Marquette, in the Baraboo country, and to the northeast from there, we have a great development of the porphyry so characteristic of the Huronian iron district of Missouri. It is wholly within the possibilities that iron ore deposits may yet be discovered in the Baraboo rocks.

Iron Ores associated with the Potsdam Sandstone.—In the counties lying immediately north of the Wisconsin River, along its final westward stretch to the Mississippi, certain layers of the Potsdam are very highly charged with red hematite, and this at times nearly excludes the sand, yielding a fair ore. A very good red hematitic ore occupying such a position has been opened on in the eastern part of the town of Westfield, T. 11, R. 4 E., but the deposit has not yet been at all developed. At several points in the western part of Sauk

County, and again farther west, in Richland and Crawford counties, valuable ores occur in connection with the Potsdam sandstone. The best known deposits, both of which are now being worked, and at each of which there is a small blast furnace, are near Ironton, in Sauk County (T. 12, R. 3 E.), and at Cazenovia, in Richland County (T. 12, R. 2 E.). At both of these points the ore, which is brown mingled more or less with red hematite, fills in a very irregular manner crevices and openings between more or less detached masses of the Potsdam sandstone.

Clinton Iron Ore.—The well-known fossil or "dyestone" ore occurring at so many points in the Eastern States at the Clinton horizon, has also a large development in Wisconsin. The Clinton formation in the region of Lake Michigan merges into the great mass of limestone which forms nearly the whole of the Upper Silurian; but at the junction of this limestone mass with the underlying Cincinnati shales the Clinton ore has been found at several points. This junction line is indicated closely on the accompanying map. Professor Chamberlin has shown that the ore certainly does not form a continuous band at the junction, though always occupying this position when found. Though not improbably existing at many points now unknown, it has as yet been developed in quantity at only two points. At these, however, the deposit is of enormous dimensions. At Iron Ridge, in the town of Hubbard (T. 11, R. 11 E.), the ore is found with a thickness of 15 to 25 feet, underneath a west-facing ledge of the Niagara limestone. It is in more or less distinct layers, three to fourteen inches in thickness, most of which are made of lens-shaped concretionary grains, $\frac{1}{25}$th inch in diameter. These grains are often so little coherent that the ore is shovelled out like sand. Some of the lower portions are more or less hydrated, but the mass is ordinary red hematite. At the Mayville deposits, four miles north, there is a large mass of loose ore in the overlying glacial drift, evidently derived from the ore-bed itself, so that the whole thickness of the deposit here is as much as 40 feet. Two small charcoal furnaces, one at Mayville and one at Iron Ridge, smelt this ore on the ground, producing an iron exceedingly rich in phosphorus, and using either no flux, or sandstone, there being a large proportion of lime carbonate and clayey matter contained in the ore. The average furnace yield is 45 per cent. Much the larger part of the ore, however, is sent away to mingle with Lake Superior and Missouri Huronian ores, especially the former. It goes to Chicago, Joliet, and Springfield, Illinois; St. Louis, Missouri; Wyandotte and Jackson, Michigan; and Appleton, Green Bay, and Milwaukee,

Wisconsin. The Iron Ridge mine produced in 1872, 82,371 tons. The great extent of these deposits, their great accessibility, from occurrence on the side of a ridge and in a thickly settled region, and the usefulness of the ore as an admixture for the siliceous ore of Lake Superior, give it a very great value, notwithstanding the large content of phosphorus.

Bog Ore.—In Central and Northern Wisconsin there are very numerous and large marsh areas, underneath many of which considerable quantities of a rich brown hematite ore are to be found. The thickness of the ores is not usually over two or three feet, and is commonly less, but it would be so easy to win, that it is not unlikely to have a considerable value in the future. Near Necedah, in Juneau County, and Grand Rapids, in Wood County, are promising deposits of this character, though yet imperfectly exposed.

Analyses.—The following table of analyses, arranged in a convenient form for comparison, will give more definite ideas as to the deposits above described. Nos. 1 to 11 inclusive are samples from the Penokee iron range, and were made by Prof. W. W. Daniells, of the University of Wisconsin, and T. B. Bowman. Nos. 1 to 5 inclusive are samples taken from the west bluff at Penokee Gap, Sec. 14, T. 44, R. 3 W.; No. 1 represented a thickness of 19 feet; 2, of 18 feet; 3, of 10 feet; 4, a 10-inch seam of granular magnetite; 5, a 2-inch rich seam. No. 6 is of a sample from the N.E. quarter of Sec. 15, T. 44, R. 3 W., representing 41 inches; No. 7, of a sample from the N.E. quarter of Sec. 14, T. 44, R. 3 W., representing 50 inches; No. 8, of one from the S.E. quarter of Sec. 10, T. 46, R. 3 W., representing 58 feet; No. 9, of one from S.W. quarter of Sec. 1, T. 44, R. 2 W., representing a thickness of 20 feet; No. 10, of one from rich seams, S.E. quarter of Sec. 32, T. 45, R. 1 W.; and No. 11, of one representing a thickness of 25 feet, from near the Potato River, Sec. 19, T. 45, R. 1 E. Nos. 12 to 19 inclusive represent samples of the Menominee Huronian ores, and are taken from C. E. Wright's report on that region. No. 12 represents the 36-feet bed; No. 13, an overlying 10 feet of leaner ore; and No. 14, the 68-feet bed—all from the Commonwealth mine. No. 15 represents 10 feet of specular ore; No. 16, 15 feet of hard hematite; No. 17, 18 feet of soft hematite and limonite; No. 18, 9 feet of soft hematite; and No. 19, 6 feet of shaly specular ore—all from the Eagle mine. No. 20, by Chilton, represents the Clinton ore of Iron Ridge, Dodge County. Nos. 21 and 22, by Oliver Matthews, represent bog ores, from Necedah, Juneau County, and Grand Rapids, Wood County, respectively.

Analyses of Wisconsin Iron Ores.

	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
Metallic iron...........	44.941	49.40	45.871	57.520	68.042	41.192	53.465	41.934	37.885	67.665	35.449	63.182	48.10	57.50	49.00	53.03	57.00	58.10	58.10	55.475	49.98	51.26
Protoxide of iron.....	19.173		19.479	24.602		8.460	8.382	16.797	12.331	27.488	5.112											
Sesquioxide of iron..	42.897		43 885	54 825		49.435	67.064	41.241	40.420	65.913	49.254	90.26								79.25	71.40	73.23
Silica........................	31.838	27.03	30.734	17.276		33.894	18.472	36.508	39.171	4.684	40.103	3.24	10.60	5.57	20.30	11.60	3.80	3.04	4.00	6.18	8 52	4.81
Organic matter........																					1.62	5.88
Alumina..................	0.384		none	none		1.151	0.305	1.025	1.139	none	2.059									2.49	3.77	1.00
Lime.........................	1.373		1.910	2.043		3.156	2.483	1.383	1.373	1.786	1.687									* 6 81	.58	0.11
Magnesia.................	1.293		1.632	0.660		2.403	2.280	2.156	1.890	none	1.586									† 0.14	trace	0.25
Manganese oxide.....	1.126		0.873	0.625		0.337	1.050	0.193	0.553	0.563	0.183										0 27	
Phosphoric acid.......	none	none	0.021	trace		none	0.127	trace	trace	none	none	0.53	‡ .15	‡ .21	‡ .14		‡ .40		‡ .39	1.13	0.21	0.10
Sulphur....................	none		none	trace		none	none	0.160	none	trace	0.199	0.02									0.02	0.07
Water......................	0.378		0.545	0.282		1.500	0.450	1.078	2.559		trace									4.00	13.46	14.24
Totals...........	98.462		99.079	100.313		100.336	100.613	100.541	99.436	100.434	100.183									100.00	99.85	99.69
Magnetic oxide........	50.668		62.754	79.273		27.260	27.008		39.733	88.572												
Specular oxide.........	11.402		0.610	0.154		30.635	48.438		13.018	4.829												
Total.............	62.070		63.364	79.427		57.895	75.446		52.751	93.401												

* Calcium carbonate. † Magnesium carbonate. ‡ Phosphorus.

LEAD AND ZINC.

Lead has for many years been the chief metallic production of Wisconsin, and, together with zinc, whose ores have only been utilized since 1860, still holds this prominent position, although the production is not nearly so great as formerly. The entire product of these metals in Wisconsin comes from that portion of the extreme southwestern part of the State which lies west of Sugar River, and south of the nearly east and west ridge that forms the southern side of the Valley of the Wisconsin from the head of Sugar River westward. This district is commonly known as the Lead Region, or, more generally, as the Upper Mississippi Lead Region, including the adjoining portions of Iowa and Illinois.

What European first became acquainted with the deposits of lead in the upper portion of the Valley of the Mississippi is a matter of some doubt. Charlevoix (*Histoire de la Nouvelle France*, iii, 397, 398) attributes the discovery to Nicolas Perrot, about 1692, and states that in 1721 the deposits still bore Perrot's name. Perrot himself, however, in the only one of his writings that remains, makes no mention of the matter. The itinerary of Le Sueur's voyage up the Mississippi, 1700-1701, given in La Harpe's history of Louisiana, which was written early in the eighteenth century, shows that the former found lead on the banks of the Mississippi, not far from the present southern boundary of Wisconsin, August 25th, 1700. Captain Jonathan Carver, 1766, found lead in abundance at the Blue Mounds, and found the Indians in all the country around in possession of masses of galena, which they had obtained as "float mineral," and which they were apparently incapable of putting to any use. There is no evidence of any mining before Julien Dubuque, who (1788 to 1809) mined in the vicinity of the flourishing town which now bears his name. After his death in 1809 nothing more was done until 1821, when the attention of Americans was first drawn to the rich lead deposits of this region. By 1827 the mining had become quite general, and has continued to the present time, the maximum production having been reached, however, between the years 1845 and 1847.

As is well known to geologists, the lead of the Upper Mississippi region occurs in more or less water-worn, and often greatly enlarged, limited crevices, for the most part joint cracks, in the wholly undisturbed limestone layers of the Trenton group. The complete series of rocks of the lead region aggregates some 2000 feet in thickness, of nearly horizontal strata, piled upon an irregular floor of Archæan

gneiss and schists. The following tabulation shows the relations of these layers and the position of the lead horizon:

	Formation.	Thicknesses.
	Niagara dolomitic limestone, . .	300 feet.
	Cincinnati shales, ...	60- 100 "
Lead Horizon,	Galena dolomitic limestone, . .	250- 275 "
	Blue limestone, ...	50- 75 "
	Buff dolomitic limestone, . . .	15- 20 "
	Lower magnesian (dolomitic) limestone,	250 "
	Potsdam sandstone series, . . .	800-1000 "

Lead has also been found in small crevices in the Lower Magnesian at points outside the limits of the productive lead region, within which it is only rarely reached by the deeper channels of erosion. Whitney, in his well-known report, expresses his disbelief, from what can be seen of the Lower Magnesian in the immediate vicinity of the lead region, in its ore-bearing character. Although this may very possibly be an incorrect opinion, since it is not based upon investigation of the formation within the productive lead region, it seems evident, as Whitney says, that the crevices of the Lower Magnesian, if such exist, being wholly independent of those of the Trenton and Galena above, and separated from them by over a hundred feet of barren sandstone, can be only sought for by sinking at random through the overlying formations.

From the above table it will be seen that the total thickness of the lead horizon is only very rarely left undenuded. Single crevices, moreover, never extend through more than a small portion of the mining ground. Zinc and lead are found in the same kind of deposits, being frequently raised from the same openings. Much the larger part of the zinc ores, however, comes from the Blue and Buff limestones, and the lowest layers of the Galena, while the lead ores, though obtained throughout the whole thickness of the mining ground, are especially abundant in the middle and upper layers of the Galena. The ore occurs almost wholly as Galena, though small quantities of the earthy carbonate are found as float mineral. The zinc occurs equally as black-jack (sphalerite) and as dry-bone (smithsonite), both ores being highly ferruginous.

The ore deposits are of two general kinds, which may be distinguished as vertical crevices and flat crevices, the former being much the more common. The simplest form of the vertical crevice is a narrow crack in the rock, having a width of a few inches, an extension laterally of a few yards to several hundred feet, and a verti-

cal height of 20 to 40 feet, thinning out to nothing in all directions, and filled from side to side with highly crystalline, brilliant, large-surfaced Galena, which has no accompanying metallic mineral or gangue matter. Occasionally the vertical extension exceeds a hundred feet, and sometimes a number of these sheets are close together and can be mined as one. Much more commonly the vertical crevice shows irregular expansions, which are sometimes large caves, or openings in certain layers, the crevice between retaining its normal character, while in other cases the expansion affects the whole crevice, occasionally widening it throughout into one large opening. These openings are rarely entirely filled, and commonly contain a loose, disintegrated rock, in which the Galena lies loose in large masses, though often adhering to the sides of the cavity in stalactites, or in cubical crystals. The vertical crevices show a very distinct arrangement parallel with one another, there being two systems, which roughly trend east and west, and north and south. The east and west crevices are far the most abundant and most productive of ore. The vertical crevices are confined nearly altogether to the upper and middle portions of the Galena, and do not yield any quantity of zinc ores. Of 2232 crevices surveyed by Mr. James Wilson, Jr., in 1877, 1327 were east and west, 482 north and south, 322 quartering, and 51 irregular. The crevices are gathered into upwards of 40 distinct groups, of which about 28 are producing. The amount raised from single crevices has often been over a million pounds.

The zinc ores were formerly rejected as useless, and have only been utilized since 1860, just before which time Professor Whitney had expressed in his report the opinion that they would never be found in quantity sufficient to pay. An attempt to smelt them at Mineral Point was not successful, because the amount of fuel and clay needed, both of which have to come from a distance, exceeding even the amount of ore used, caused a very heavy expense for transportation. The ores are, therefore, now taken altogether to La Salle and other points in Illinois, where they meet the fuel and clay, and the industry at that place has become a flourishing one. The amount of zinc ore in the Wisconsin lead region is beyond doubt very great, and will be a source of wealth for a long time to come. At the present time the annual production of the lead region is from 10,000,000 to 12,000,000 pounds galena, 18,000,000 to 20,000,000 pounds of dry bone, and 15,000,000 pounds of black-jack. One of the principal districts is in the vicinity of Mineral Point, which is also the point of shipment for nearly all the zinc ore. There are two

lead furnaces at this point, which, in the ten years preceding 1873, produced 23,903,260 pounds of lead, or an annual average of 1,991,938 pounds, the maximum being, in 1869, 2,532,710 pounds; the minimum, in 1873, 1,518,888 pounds.

In conclusion we may refer to the admirable contour-line and geological maps of the lead region by the late Mr. Strong. By the aid of these maps it is possible to know the exact extent of the mining ground underneath every point of the district.

COPPER.

As already indicated, the copper-bearing series of Lake Superior has an immense surface spread in Northern Wisconsin. Native copper has been observed at a large number of points, scattered all along the course of the formation to the Minnesota boundary. Several attempts at mining were made before the war, but were never carried far enough to prove the deposits. The principal workings were on the Montreal and Bad rivers, in Ashland County, and Brulé, Aminicon, and Black rivers, in Douglas County. On the Montreal and Bad the copper was sought in veins, but on both rivers the workings disclosed belts of an amygdaloid which had undergone the laumonitic decay, in which minute particles of copper were quite thickly disseminated. In Douglas County the copper was found in so-called longitudinal veins, which are really belts of highly altered amygdaloid, that have undergone the more common epidote-quartz decay, the copper being found in a matrix composed of those minerals and calcite. Mr. Strong has found native copper at a number of points about the headwaters of the St. Croix and Numakagon rivers, and some has been taken out as far southwest as St. Croix Falls. In all of this region the characters of the system as developed in the typical region of Keweenaw Point hold well, and there is certainly at least sufficient encouragement to warrant careful explorations by land-owners.

Small lots of much rotted but quite rich copper pyrites have been taken out of limited crevices in the Galena limestone at a few points in the lead region of the southwestern part of the State, and also just north in the Potsdam sandstone of Richland and Vernon counties, but there is no present prospect that any large quantities will be discovered in the future.

SILVER.

The peculiar quartzless sandstone, and accompanying shale, which carry the silver of the Ontonagon or Iron River mines of Michigan

continues, with the same characters, across the Montreal River into Wisconsin. I have detected traces of silver in samples of the rock from the Montreal.

BRICK CLAYS.

These constitute a very important resource in Wisconsin. Extending inland for many miles from the shores of Lakes Michigan and Superior are stratified beds of clay of lacustrine origin, having been deposited by the lakes when greatly expanded beyond their present sizes. All of these clays are characterized by the presence of a large amount of carbonate of lime. Along Lake Superior they have not yet been utilized, but all through the belt of country bordering Lake Michigan they are dug and burned, fully 50,000,000 bricks being made annually in this region. A large proportion of these bricks are white or cream-colored, and these are widely known under the name of Milwaukee brick, though by no means altogether made at Milwaukee. Others are ordinary red brick. This difference in color is usually attributed to the greater amount of iron in the clay from which the red bricks are burned, but it has been shown by Sweet that the white kinds are burned from clay which often contains more iron than that from which the red bricks are made, but which also holds a very large amount of carbonate of lime. The following analyses show (1) the composition of the clay from which the cream-colored bricks are burned at Milwaukee, (2) the composition of a red-brick clay from near Madison, and (3) that of the unutilized clay from Ashland, Lake Superior. Nos. 1 and 2 are by Sweet, No. 3 by Bowman.

	1	2	3
Silica	38 22	75.80	58.08
Alumina	9.75	11.07	25.38
Iron sesquioxide	2.84	3.53	4.44
Iron protoxide	1.16	0.31	
Lime	16.23	1 84	8.30
Magnesia	7.54	0.08	
Carbon dioxide	18.50	1.09	
Potash	2.16	1.74	
Soda	0.65	0.40	
Water	0.95	1.54	4.09
Moisture	1.85	2.16	
Total	99.85	99.56	100.29

The white bricks are burned at a very high temperature, and lose their color only at incipient vitrification. The change seems to be in the nature of a production of a lime-iron silicate. Ordinary red

clay burned with oyster-shells may be made to yield the same result. At Milwaukee 24,000,000 cream-colored bricks are made annually; at Racine, 3,500,000; at Appleton and Menasha, 1,800,000 each; at Neenah, 1,600,000; at Clifton, 1,700,000; at Waterloo, 1,600,000; and in smaller quantities at Jefferson, Fort Atkinson, Edgerton, Whitewater, Geneva, Ozaukee, Sheboygan Falls, Manitowoc, Kewaunee, and other places.

Although these lacustrine clays are much the most important in Wisconsin, excellent brick clays are also found in the interior of the State. In numbers of places along the Catfish Valley, in Dane County, excellent stratified clay occurs. At Madison this is burned to a red brick; at Stoughton and Oregon to a fine cream-colored brick. At Platteville, Lancaster, and other points in the southwestern part of the State, red bricks are made from residuary clays found in the vicinity.

KAOLIN.

The existence of kaolin in Wisconsin has been known for many years. The material has, however, only very recently attracted much attention and become the object of actual exploitation.

The various localities at which kaolin has been noticed in the State, so far as my knowledge extends, all occur in a belt of country about fifty miles in length and fifteen in breadth, stretching eastward from Black River, in Jackson County, to the Wisconsin, in the vicinity of Grand Rapids, in Wood County. This district includes more or less of townships 21, 22, and 23 north, and ranges 1, 2, 3, 4, west, and 1, 2, 3, 4, 5, 6, 7, east of the meridian. It is crossed from north to south by three streams of considerable size: Black River on the west, the Wisconsin on the east, and Yellow River towards the centre. The kaolin discoveries have, I believe, been made almost entirely in the vicinity of these streams.

The district thus described lies, for the most of its extent, just south of the main boundary line between the Potsdam sandstone, which underlies such a large area to the south, east, and west, and the Laurentian gneiss. The country in this part of the State is generally level, with a gradual rise to the northward. In the more southern portions of the belt the sandstone is nearly everywhere the surface rock, except along the beds of rivers, where the gneiss and granite are laid bare. The sandstone is, therefore, where it occurs, only a very thin covering over the crystalline rocks, and, indeed, these oc-

casionally rise through it in bold isolated bluffs of granite and quartzite. Further north, the gradual rise of the country seems to be due in some measure to the shape of the surface of the underlying gneiss, which finally rises from beneath the sandstone and becomes the surface formation. A geological map, including Portage, Wood, Clark, and Jackson counties, would show on the south the sandstone as the surface formation, on the north the crystalline rocks, while where the two meet they would be shown dovetailing into each other, the gneiss extending many miles south in the stream beds, the sandstone penetrating as far north on the divides. As we trace the rivers southward towards where the last crystalline rocks are seen, these are found confining themselves more and more closely to the vicinity of the streams until they are finally restricted to their beds, the sandstone forming the banks. Thus the Wisconsin River for ten miles above Point Bass, and the Black for a greater distance above the falls, present strips of crystalline rocks only as wide as their own currents. The boundary line between the driftless area of the southwestern quarter of the State and the drift-bearing region to the north and east, crosses the district in a nearly east and west line from Grand Rapids to Black River Station, on Black River.

The kaolin occurs entirely as "kaolinized" rock. It has been noticed only in the vicinity of the large streams, because elsewhere the crystalline rocks are for the most part covered by sandstone. Nearly always it occupies exactly the original position, retaining sometimes the minute structure of the unaltered rock. The rocks from which the kaolin has been formed, and into which it can frequently be traced through every degree of alteration, are beds interstratified with the series of Laurentian strata, which have, over wide areas, a common strike. Only the outcropping edges of these beds are decomposed, and as a consequence it follows that the resulting kaolin tends to form narrow bands crossing the country in straight lines parallel to the general strike. It is common to find overlying the kaolin a few layers of sandstone, sometimes a few inches only; at others a score or so of feet. In such cases the purer kaolin is found immediately below the sandstone, next below a partially kaolinized rock, and next below again the entirely unaltered rock.

The kaolin appears to be almost entirely within the driftless area, or at least where the drift is very thin, and the glacial action has been slight. This fact becomes a significant one when we consider that over all the great gneiss region of the northern half of the State, which is drift-covered, no occurrence of kaolin is known; all

Analyses of Wisconsin Kaolins.

	1	2	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
Silica	78.83	49.94								70.83				70.25		69.34				54.87				
Alumina	13.43	36.80								18.98				17.68		19.19				28.87				
Sesquioxide of iron	0.74	.72	1.69	2.30						1.24		2.30		2.32		1.75	1.95			1.54				
Lime	0.64	trace								0.24				0.33		0.44				1.62				
Magnesia	0.07									0.02				1.49		0.31				.99				
Potash	.37	.51			.78		0.38	1.21	.87	2.49	1.22	2.30	1.96	1.69	2.33	3.30	1.84	2.65	2.95	2.57	1.25	2.18	1.51	1.54
Soda	.07	.08			.03		0.08	0.46	.00	0.10	trace	trace	0.05	0.39	0.10	2.43	0 27	.21	.83	.07	0.08		0.81	0.22
Water	5.45	11.62								5.45	8.84			5.61		2.67		7.29		9.48				8.69
Protoxide of iron																				.95				
Carbon-dioxide	.01									0.02				trace										
Total	99.61	99.67								99.37				99.76		99.43				100.96				
Coarse sand	67.30					42.59													56.61					
Fine clay	32.70					57.41													43.39					
Total	100.00					100.00													100.00					

the known occurrences being confined to that comparatively small district where the crystalline rocks are found within the driftless area. This absence of kaolinized rock in the northern portion of the State is evidently due to the denuding agency of the drift forces.

The best known kaolin deposits in Wisconsin are those that occur on and near the Wisconsin River, in the vicinity of Grand Rapids, in Wood County. The gneissic rocks here occur chiefly in the bed of the stream, which, for many miles, makes bold rapids over their upturned edges. Elsewhere they are mostly covered with sandstone. The predominating gneissic rocks have associated with them both interbedded and clearly intrusive granite, besides hornblendic schists and intrusive diabase. Of the gneiss and granite there are many varieties, according to the predominance of one or other mineral ingredient; both rocks being formed sometimes of a largely preponderating pinkish feldspar. These beds are the ones most commonly weathered, though some of the dark, micaceous kinds show the same tendency. All of the beds strike between N. 50° E. and N. 80° E., with a dip of about 50°, either SE. or NW. The occurrences on Yellow and Black rivers are very similar.

Taking the whole district together a very large amount of kaolin undoubtedly exists. It must always, however, be expected that any one deposit will vary much in character, both as to purity and as to thickness. Numbers of instances came to my notice where borings showed two feet of kaolin and no kaolin at all, within a few feet of one another. The fact that kaolin is apt to occur in continuous lines will, however, counterbalance the disadvantage of its lack of uniformity, since it can be searched for with assurance of success. In my opinion the indications are such as would warrant the outlay of money in exploring the deposits and testing more completely on the large scale their refractoriness. The following are analyses of Wisconsin kaolins by Sweet:

Nos. 1, 4, 5, 6, 7, 9, 11, 13, 15, 18, 22, 24 are analyses of samples of crude clay, dried at 100°, from a number of different openings, and represent a thickness of from one to four feet. All of the others, except 17, which is an only partly kaolinized gneiss, represent levigation products from the foregoing clay, Nos. 2, 8, 10, 12, 14, 16, 19, 21, 23, and 25 being the fine clay washed respectively from the crude clays 1, 7, 9, 11, 13, 15, 18, 20, 22, and 24, while 3 is the coarse residue from 1 after washing. The residue is chiefly composed of quartz grains, but also in part of fragments of mica and undecomposed feldspar.

HYDRAULIC LIMESTONES.

Certain layers of the Lower Magnesian limestone, as at Ripon and other points in the eastern part of the State, are known to produce a lime which has in some degree the hydraulic property, and the same is true of certain layers of the blue limestone of the Trenton group in the northwestern part of the State. The most valuable material of this kind, however, that is as yet known to exist in Wisconsin is found near Milwaukee, and has become very recently widely known as the "Milwaukee" cement-rock. This rock belongs to the Hamilton formation, and is found exposed near the Washington Street bridge, at Brown Deer, on the lake shore at Whitefish Bay, and at other points in the immediate vicinity of Milwaukee. The quantity attainable is large, and a very elaborate series of tests by D. J. Whittemore, chief engineer of the Milwaukee and St. Paul Railroad, shows that the cement made from it exceeds all native and foreign cements in strength except the famous English Portland cement. The following are three analyses of the rock from different points, and they show that it has a very constant composition:

	1	2	3
Calcium carbonate	45.54	48.29	41.34
Magnesium carbonate	32.46	29.19	34.88
Silica	17.56	17.56	16.99
Alumina	1 41	1.40	5.00
Iron sesquioxide	3 03	2.24	1.79
	100.00	98.68	100.00

BUILDING-STONES.

All the rocky formations of Wisconsin are used in building, and even the briefest synopsis of the whole subject of the building-stones of the State would exceed the limit of this paper. A few of the prominent kinds alone are mentioned.

Granite occurs in protruding masses, and also grading into gneiss, in the northern portions of the State, at numerous points.

None of these granites or of the other crystalline rocks of Wisconsin have yet been utilized, but on Yellow and Black rivers, and in the intervening country, a red granite of extraordinary beauty, and of thorough durability, occurs very largely.

The handsomest and most valuable sandstone found is that which extends along the shore of Lake Superior, and which forms the base-

ment rock of the Apostle Islands. On one of these islands a very large quarry is opened, from which are taken masses of almost any size of a very close-grained, uniform, dark-brown stone, which has been shipped largely to Chicago and Milwaukee. At the latter place the well-known court-house is built of this stone. An equally good stone can be obtained from the neighboring islands, and from points on the mainland. A very good white to brown indurated sandstone is obtained from the middle portions of the Potsdam series.

All of the limestone formations of the State are quarried for building-stone. The layer known locally as the Mendota limestone, included in the upper layers of the Potsdam series, yields a very evenly bedded, yellow, fine-grained rock, which is largely quarried along the Valley of the Lower Wisconsin, and also in the country about Madison. In the town of Westport, Dane County, a handsome, fine-grained, cream-colored limestone is obtained from the Lower Magnesian. The Trenton limestone yields an evenly bedded, thin stone, which is frequently used for laying in-walls. The Galena and Niagara are also utilized, and the latter is capable, in much of the eastern part of the State, of furnishing a durable, easily dressed, compact, white stone.

UNIVERSITY OF WISCONSIN, MADISON,
February, 1880.

THE DETERMINATION OF SILICON AND TITANIUM IN PIG IRON AND STEEL.

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IN a communication to this Institute at the Baltimore meeting, February, 1879,* on the "Determination of Silicon in Pig Iron and Steel," the method recommended was the treatment of the metal with nitric acid until action had ceased, and then evaporating with sulphuric acid until the nitric acid was nearly or quite driven off. After filtration of the siliceous and carbonaceous residue, and washing with hot water and hydrochloric acid, a silica was obtained, on ignition, which was quite pure. Since this paper was read before the Institute, we have had large experience with the method, and

* Transactions, vol. vii, p. 346.

find it uniformly reliable. Results are obtained in a few hours with the greatest accuracy.

While testing the method and comparing it with others in general use, and also with some new methods, there have been developed some facts which may be of sufficient value to lay before the members of the Institute.

Interesting results were obtained by the treatment of iron borings in a platinum crucible with acid potassium sulphate at a red heat. The operation must be conducted with care, to prevent too violent action; but a little practice will enable one to effect the complete oxidation of one gram of iron (the amount usually taken) in from 20 to 30 minutes. On subsequent solution of the fused mass in water, a little hydrochloric acid is added to dissolve any ferric oxide which may adhere to the crucible. For 1 gram of iron, about 25 grams of the acid potassium sulphate are used. This amount is ordinarly added at once to the iron in the crucible. The operation must, of course, be carefully watched that the mass does not flow over the top. It should not mount higher than three-fourths of the height of the crucible, which should have a capacity of not less than 70 cc. If the operation has been successful, a nearly white mass will remain in the crucible, without a particle of graphite. The mass may be poured out while liquid, but a more convenient method is to insert into the fluid mass a piece of heavy platinum wire, bent at the end, and then allow the mass to solidify around it. The crucible is then slightly warmed to loosen the contents, which can be lifted out by the wire. The fused mass, with the crucible and lid, is put at once into boiling water with some hydrochloric acid. When solution is complete, the silica is filtered off and washed with hot dilute hydrochloric acid and water. After drying, the filter, with its contents, is ignited and weighed. The resulting product should be pure white. While accurate results have been obtained by this method in 45 minutes, yet a long experience with it shows that it is not to be relied on for all kinds of iron and steel. The following are some of the results obtained:

	1	2	3	4	5	6	7	8	9
Silicon by nitric and sulphuric acids	0.737 0.739	0.772 0.777	1.24 1.25	1.214	2.35 2.37	2.40 2.42	2.40 2.40	0.755 0.757	0.622 0.629
Silicon by fusion with acid potassium sulphate	0.628 0.677	0.772 0.702 0.677	1.41	1.250	2.37	2.53	2.46 2.47 2.47 2.49 2.50 2.50 2.52 2.52 2.54 2.58 2.69	0.749 0.761	0.535 0.566 0.575 0.591

	10	11	12	13	14	15	16	17	18
Silicon by nitric and sulphuric acids	1.92	0.207 0.208 0.210	4.40 4.39	0.165 0.166	0.929	3.75	0.027 0.027	0.682 0.663	0.205 0.207
Silicon by fusion with acid potassium sulphate	1.94 1.94	0.093 0.141 0.231 0.287	4.60	0.116 0.130	0.929	3.94 4.01 4.02 4.19	0.000 0.025	0.529 0.563 0.598 0.628 0.661	0.065 0.115 0.120 0.197 0.209

1. Richmond warm-blast charcoal iron, No. 3. 2. Greenwood cold-blast charcoal, No. 1. 3. Dutchess, anthracite, No. 1. 4. Hecla cold-blast charcoal, No. 2. 5. Bushong, anthracite, No. 1. 6. Leesport, anthracite, No. 1. 7. South Easton, anthracite, No. 1. 8. Glendon, gray forge. 9. Glendon, mottled. 10. Durham, anthracite. 11. White iron. 12. Silver-gray iron. 13. Spiegeleisen. 14. Source unknown. 15. Source unknown. 16. Bessemer steel. 17. Bessemer steel. 18. Sanderson tool steel.

In the above table will be noticed many results which vary greatly from the true percentage, and for which variation no sufficent explanation is at hand. In general, it may be said that irons high in silicon give better results than those low in silicon. With silicon over one per cent., the tendency is toward too high results; with silicon under one per cent., the tendency is toward low results. When the silicon is about one-half of one per cent. or lower, the results are, moreover, very uncertain, as will be seen from the figures for mottled and white iron, also for spiegeleisen and steel. In one experiment on a sample of Bessemer steel (No. 16), no silicon was found, while, in another experiment with the same steel, made by completely driving off the free sulphuric acid from the acid sulphate, and then adding a fresh portion, the percentage of silicon obtained agreed with that by nitric and sulphuric acids. For iron or steel very low in silicon, this last procedure is necessary to get even approximate results; but for ordinary pig irons, it gave no better results than were obtained by simply heating the borings with acid potassium sulphate until all traces of graphite had disappeared.

Silver-gray iron is with difficulty oxidized by this method, although the results obtained from one sample were reasonably good.

For Bessemer works, where a rapid method for the determination of silicon is often desirable, this method will perhaps find a useful application. It should be mentioned that we have found great difficulty in buying acid potassium sulphate free from silica or other insoluble matter. In all cases, we found it necessary to purify the sulphate by solution in water, filtration, evaporation, and fusion.

Some variations were tried on the method. The pig iron was first oxidized in the crucible by nitric acid and the resulting product treated with the acid sulphate. Again, nitre was used in connection with the acid sulphate. In another series of experiments, the iron was heated to redness for some time with sodium carbonate (which has the effect of oxidizing energetically the carbon and silicon),* and subsequently treated with sulphuric acid and acid sulphate. These variations were not accompanied with any better results than when the acid sulphate was alone used.

The high results are mostly caused by oxide of iron, which attaches itself in small amount to the upper part of the crucible, and which is somewhat slow of solution in acid. It does not follow that silica which is quite white after ignition is free from iron.

The facility with which pig iron and steel can be brought into complete solution by fusion with acid potassium sulphate will perhaps recommend this procedure when other ingredients besides silicon are to be determined.

In comparing the silicon results obtained by the nitric and sulphuric acid process with those obtained by the use of hydrochloric acid, we noticed that the results by the latter process were almost invariably higher when the residual silica obtained after burning off the carbon was not refused with alkaline carbonates. The same is true when sulphuric acid is used alone without nitric.

In many cases, the silica was found to contain iron oxide or other bases, but the higher results were also obtained when the silica was found to be free from metallic oxides. Investigations showed the presence of titanic acid, and an extended series of experiments has shown that titanium is very generally present in pig iron.

In determining the titanium, Riley's method was generally used, which consists in treating the pig iron with hydrochloric acid and

* Transactions, vol. vii, p. 146.

filtering off the siliceous graphitic residue, which is, after ignition, fused with acid potassium sulphate. This method gives fair results, but a more accurate method we found to be the treatment of pig iron in a porcelain boat in a glass tube with dry chlorine at a red heat. Pig iron thus treated is almost completely volatilized, a small carbonaceous residue—five per cent. or less—remaining in the boat. The ferric chloride, with some manganic chloride, condenses in the glass tube (which should be long enough to allow of this), and the non-metals are driven over as gaseous chlorides.

For the absorption of the silicon and titanium, a series of three or four tubes or bottles of water is used. No precipitate is noticed in the water, but, on boiling, titanic acid contaminated with silica is precipitated. To determine the silica and titanic acid, the contents of the bottles are poured into an evaporating dish and strongly acidified with hydrochloric acid. Fifteen cubic centimeters of sulphuric acid (sp. gr. 1.23) are added, and the solution evaporated until all the hydrochloric acid is expelled. The silica is thus rendered insoluble and the titanic acid retained in solution, from which it can be precipitated, after dilution, by boiling. The results by this method are always a little higher than those obtained by Riley's method. In the treatment of pig iron by nitric and sulphuric acids, the silica obtained is free from titanic acid, which goes entirely into the filtrate. It is not possible, however, to get more than about one-third of the total amount on precipitation by boiling, owing, doubtless, to the presence of the relatively large amount of iron in solution.

The following table shows the relation between the silicon and titanium in a few pig irons containing notable quantities of titanium:

	Glendon gray-forge.	Silver-gray.	Source unknown.	Source unknown.	Leesport.	Bushong.
Titanium by Riley's method...	0.099	0.114	....	0.318	0.115	0.225
Titanium by chlorine method.	....	0.278	0.216	0.374	....	....
Titanium calculated as silicon.*	0.077	0.217	0.170	0.291	0.081	0.173
Sum of last with true percentage of silicon..................	0.832	4.607	1.460	1.751	2.481	2.523
Silicon by HCl method, without refusing........................	0.811 0.815	4.650	1.640	1.840	2.520 2.550	2.590 2.580
True percentage of silicon......	0.755	4.390	1.290	1.460	2.400	2.350

* That is, the amount of silicon which would be calculated from the titanic acid mixed with the silica resulting from the hydrochloric acid treatment.

Other determinations of titanium in pig iron by Riley's method are as follows:

	Per cent. of Titanium.
Richmond, warm-blast, charcoal, No. 3,	0 018
Greenwood, cold-blast, charcoal, No. 1,	0.052
Hecla, cold-blast, charcoal, No 2,	0 048
Dutchess, anthracite, No. 1,	0.055
Leesport, anthracite, No. 1,	0.115

A few more details of the treatment of pig iron with dry chlorine may be worth giving in the accompanying tabular form:

PIG IRON TREATED.	Total percentage of residue in boat.	Siliceous residue after burning off carbon.	Percentage of carbon thus determined.	Silicon determined in this residue.	Silicon from solution in water.	Total silicon by Cl. method.	Total silicon by HNO_3 and H_2SO_4 process.
South Easton	3.92	0 292	3.628	0 031	2.335	2.366	2.400
Glendon Gray Forge.	4.17	0 200	3.970	0.022	....	....	0.756
Glendon Mottled	4.29	0.302	3.988	0.015	0.444	0.459	0.622
" "	4.15	0.266	3.884	....	....	....	0.622
Source unknown (1)	4.53	0.226	4.304	0 032	0.873	0 905	0.970
" "	4.64	0.277	4 363	0.043	0.908	0.951	....
Source unknown (2)	4.75	0.415	4.335	0.083	1.260	1 343	1.460
" "	4.74	....	....	0.080	1.340	1.420	......
" "	5.22	......	....	0.080	1.330	1.410	......
" "	4.83	0.483	4.497	0.080	1.300	1.380	....
White iron	4.04	0.224	3.816	0.080	0.152	0.232	0.209
Silver-gray	3.38	0.240	3.140	0.045	4.090	4.135	4.400

It will be seen from the above that the silicon is fairly accounted for in nearly all instances. A more thorough absorption of the silicon chloride by water, or, perhaps still better, by an alkaline solution, may give the full amount of silicon. As far as experiments go, there is no silicon with the condensed ferric chloride in the tube. Phosphorus is present in the ferric chloride, and sulphur is present as sulphuric acid in the water used for absorption, but we have not yet followed up these elements.

When the carbonaceous and siliceous residue in the boat is treated with water, a portion goes into solution, and in this solution may be detected, besides manganese, which we might expect, aluminium, magnesium, and calcium. Whence come these latter metals? Were they present in combination with the iron, or do they simply indicate the presence of cinder in the iron?

In the portion of the residue insoluble in water, these elements are likewise found, and it may be that the soluble calcium, magnesium, etc., were present alloyed with the iron, and the insoluble compounds of these metals were in the cinder. More experiments are needed to clear up this doubt.

In an experiment bearing on this point, dry chlorine was passed

at a red heat over cinder which had not been more than 24 hours out of the furnace, and it was found to increase in weight about 3 per cent., showing that the absorption of chlorine was not very marked. In a specimen of old cinder, the gain in weight under the same conditions was from 16 to 19 per cent. The action seemed to be the conversion of carbonates of the alkaline earths into chlorides.

When dry chlorine is passed over a mixture of a titaniferous ore and charcoal at a low red heat, titanium chloride is volatilized; but when a mixture of a blast-furnace cinder and charcoal is similarly treated, no silicon chloride is formed. It is possible, therefore, that the silicon remaining in the boat after the treatment of pig iron by chlorine may result from the presence of cinder in the iron. More experiments are needed before any decided assertion can be made on this point.

It was expected that the treatment of pig iron by chlorine at a low red heat would give a separation of iron from manganese. We were unsuccessful in effecting this separation. In all cases, some manganese was found with the ferric chloride. With spiegeleisen or ferromanganese, the fusion of the manganic chloride in the boat rendered it difficult to volatilize all the iron.

In the paper previously alluded to on the determination of silicon, it was stated that in the treatment of pig iron by hydrochloric acid, about one-third of the silicon was found in solution and two-thirds in the residue. Further experiments have shown that the relative amounts of silicon in solution and in the residue depend on the strength of the hydrochloric acid. Thus, in an iron containing 0.738 per cent. of silicon we found in the insoluble residue, after treating with hydrochloric acid, as follows:

With acid of sp. gr. 1.20,	0.616
" " 1.12,	0.440
" " 1.015,	0.006

Again in an iron with 2.36 per cent. of silicon we found:

With acid of sp. gr. 1.20,	2.26
" " 1.12,	2.05
" " 1.015,	0.02

There is no loss of silicon by volatilization in treating gray or white iron with hydrochloric acid.

*THE AMERICAN BLOOMARY PROCESS FOR MAKING IRON DIRECT FROM THE ORE.**

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THE direct process for the manufacture of iron which is principally used in the United States, in New York and New Jersey, is called the Jersey forge, the Champlain forge, the Catalan forge, the Bloomary forge or fire, and sometimes a Forge fire.

Dr. T. Sterry Hunt, who has discussed the process as applied to the beach sands of Canada as well as to ores, in the Report of the Canadian Geological Survey for 1867–69, has shown that the name of Catalan forge is incorrect, as this process in no way resembles the Catalan, but is really the old German bloomary modified by long use in this country. Dr. Hunt gives the probable history of the introduction of the process into the United States, and a series of historic citations of great interest relating to it. Notwithstanding the improvements which have been made in the construction of the furnace and its greater output, the process to-day, in all its essential features, is carried out as Karsten described it.

It should now be known as the American Bloomary. The name of Jersey or Champlain forge or fire is not sufficiently distinctive. The name of Bloomary, by which the process is usually distinguished, is objectionable, not because it is incorrect but because it is indefinite, as the Knobbling fires, which are a very slight modification of the Walloon process, for making blooms either from pig iron or by sinking scrap, are also called Bloomaries, and although Knobbling fires are only used in a few localities where charcoal pig is made, and special grades of charcoal-refined blooms are required, one does not, without an explanation, know whether the blooms are being manufactured from ore or pig. The name Forge fire is equally indefinite, as it might just as well apply to any of the five direct methods of making blooms which have been in use in Europe, but which have now mostly gone out of date. The name of American Bloomary, however, is the one which most completely describes it, and is the one least open to objection. The names of German, Jersey, and Champlain bloomary are too local, and these adjectives are thought unnecessary by those who use the process, and we shall not therefore describe it under any of these names, but as the American Bloomary process.

* Read at the Montreal meeting, September, 1879.

The process as it is carried on in the northern parts of New Jersey and New York, is adapted to any of the ores of iron which are rich and essentially free from impurities other than their gangue. But as the furnace is exceedingly shallow, the ore which remains a very short time in a reducing atmosphere must be usually separated from its gangue and very much enriched before it can be submitted to the operation.

To separate the gangue the ore must be calcined, crushed, and dressed. All of it is therefore in a fine state of division, in grains not larger than one-tenth of an inch in diameter. As the process is very expensive, both in labor and fuel, the loss of iron large, and as the output of the furnace is very small, it can only be applied to ores very low in phosphorus yielding a product commanding a high price. In some exceptional localities, like Crown Point, N. Y., where large quantities of ore are used for blast-furnace purposes, only those ores are treated by this process which are so poor that they would not be used in the blast furnace, so that the ore may be said to cost nothing. This is, however, an exceptional case. Most of the works mine the ore they use for the process alone, and it must therefore figure for a very large share in the items of cost.

The impurities contained in the ore are, as we have said, mostly removed by dressing; what remains is removed in the process by the formation of a silicate of iron. As silica in sufficient quantities to form the slag is generally present, except in very rare cases, no flux need be added. The temperature of the furnace is not sufficiently high to melt any slag, except a very small quantity of the silicates of protoxide of iron, and as iron oxide is present to saturate the silica, no other base need be supplied. But if the silica be present in great excess the loss in iron would be very large, so that it must be dressed out. The silica occurs as feldspar, hornblende, garnet, and quartz. Most of the ores used at Crown Point do not contain more than forty per cent. of magnetic oxide; at Au Sable Forks not more than sixty-four per cent. This is brought up to ninety-two per cent. by dressing. The beach sands contain a very variable quantity of iron and are not so easily enriched by dressing as the ore which is crushed and washed, since in addition to their gangue they contain a variable quantity of menaccanite (titanate of iron), which can only be imperfectly separated by magnetic or other separating machines. This mineral renders the sands very refractory, so that the process as applied to them has been unsuccessful on account of incomplete dressing. As usually applied it is used on

fine magnetic sands made by crushing and dressing the pure magnetic ores.

The following table gives the analyses of the dressed and raw ores used at various works in Northern New York:

	No. 1.	No. 2.	No. 3.	No. 4.	No. 5.	No. 6.	No. 7.
Magnetic oxide of iron.	95 25	64.55	92.09	94 25	96.03	73 51	94.76
Silica	3 63	30.98	5.95	4.32	3.35	23.95	4.33
Manganese oxide	0 02	0.14	0.12			0.25	trace
Lime	0 46	0.68	0.70	0.14	0.23	0.60	trace
Magnesia	0.56	0.58	0.36		0.38	0.20	trace
Alumina	0.20	1.67	0.26	0.28		1.50	
Phosphoric acid	0 076	0.13	0.16	0.038	0.014	0.04	
Sulphur	0.235	trace	trace	none			
Water	0.39	0.57	0.21	0.38			
Titanic acid	trace				trace		
	100.821	99.30	99.85	99.408	100.004	100.05	99.09

No. 1. Saranac. No. 2. Palmer ore, small lump and fine stuff (J. Blodget Britton). No. 3. Palmer ore, after separation (J. Blodget Britton). No. 4. New Bed, selected ore, (J. Blodget Britton). No. 5. Arnold Bed (O. Wuth). No. 6. Crown Point. No. 7. Chateaugay (A. H. Sherman).

The ore is usually calcined before crushing. For this purpose, at Au Sable Forks, a kiln, 26 feet long, 24 feet deep, and 12 feet high at the back, and 5 feet at the front, built of rough stones laid up with clay, is used. On the bottom, sticks of wood, 4 feet long, and 6 to 8 inches in diameter, are laid 4 feet apart, from the back to the front of the kiln. These are laid in rows, six sticks being placed in each row. Other sticks are laid above these, at right angles to them, and this wood is only from 4 to 5 inches in diameter. The wood is piled in this way until it is about 3 feet high. In the front, when they can be had, stumps of trees are piled. These cost as much as the cordwood, but they take longer to burn, and are for that reason very desirable. The partially fused ore from a previous operation is put on the wood first, then larger pieces, and so on. The fine ore is put on the top. A kiln contains about 300 tons. The best wood is used, and it takes about 25 cords of wood to a kiln. It burns out in from three to six days. As the pile sags while it is burning, and might settle together so as to completely obstruct the draft, if there were a vertical wall on the back side, a space at the back of the kiln, four feet on the bottom, and four feet high at the back, is built up solid with stone, forming an inclined plane, so that when the pile settles it slides forward, and in this way prevents sinking in a vertical line, and thus becoming packed. The object of this calcining is simply to make the ore friable.

The only fuel that can be used in the process is charcoal. None of the dense fuels, such as coke or anthracite, could be used, as they would require a greater pressure of blast, and give a higher temperature, the result of which would probably be an impure cast iron. Cast iron was formerly made in mountainous districts by a process similar to this.

Kiln coal is preferred to meiler coal, as it is free from sand and dirt, which would cause a loss of iron if present. That made from soft woods is generally used, as its reducing action is less powerful than that made from hard woods. More soft wood charcoal is required to the ton of blooms, but its use is almost universal. It is carefully sorted, so as not to contain too much fine coal, and is charged from baskets containing two bushels.

The process, as a whole, involves,

1. The manufacture of the charcoal.
2. Dressing the ore.
3. Making of blooms and billets.

The manufacture of the charcoal has been treated in another place.* The dressing will not be described here; I shall only discuss the process.

The furnace in which the ore is reduced and the bloom is made consists of a series of cast-iron plates, two to three inches in thickness, securely fastened together, forming a rectangular opening, which, at the bottom, varies from 24 to 30 inches, at right angles to the tuyere, and 27 to 32 inches parallel to it. On the back side, parallel to the tuyere, it is 28 to 36 inches high. On the front this plate is cut down to from 15 to 19 inches, to make a place for a small platform or shelf, called the fore plate. The hollow space, where the operation of reduction is carried on, is thus rectangular in shape, and is called the fire-box. Its walls are usually vertical, except the fore and skew plates, which are generally inclined, but sometimes they are made to incline outward at the rate of one inch in seven.

Each one of the plates forming the sides of the furnace has a name and a special duty. They are sometimes made of more than one piece, and are not always of exactly the same shape, nor are they always put together in exactly the same way in different works, but the variations are not very essential. Their size varies also with the

* This volume, pp. 373, 397.

capacity of the furnace, but their office is the same in all. Those plates most exposed to the direct action of the fire are usually cast with holes in them, into which pieces, called repair pieces, are made to fit, so that they can easily be removed and replaced when worn out.

Some of these repair pieces, and some of the furnace plates are water-jacketed. There are a few differences of construction in these furnaces which affect their cost, and to some slight degree the method of working. I have selected three types, distinguished by having four, three, and five hot-air pipes. See Plates I, III, and IV.*

Taking the Crown Point works at Ironville, as an example, we have the development of the furnace, shown on Plate II, Fig 13, which is the section of the furnace on the line *D D*, Plate I, Fig. 1. Fig. 13 *a*, is the hair plate or back of the furnace, cast in one piece with its section; *d* is the fore spar, which inclines from the vertical one inch in seven, whose section, with a section of the repair piece through *D D*, is shown at *e*. It is not water-jacketed. *b* is the tuyere plate, which is always vertical, shown in section *C C*, the repair piece being shown at *f*, with a section through *A A*. This is water-jacketed; the water from it flows into the bosh trough (Fig. 1, Plate 1), beside the furnace. The repair pieces in these two plates are kept from falling out by the flanges, and from falling in by two horizontal bars, one at the top and one at the bottom, which are dropped into little wrought-iron brackets bolted on to the repair piece. A cast-iron box is cast on below the tuyere-hole which is used for packing the tuyere tight with clay. *c* is the cinder plate, which inclines slightly outward, with its section. It has three rectangular holes, used for prying up the loupe, and three circular ones, which are used for tapping the cinder. The cinder plate is covered by the fore plate, upon which the loupe is drawn when it is dug up. These plates are held together by the ribs, cast on the upper side of the foundation, and the under side of the mantel-pieces (Figs. 13 and 14, Plate II; Figs. 1 and 2, Plate I), which are supported by the front and back columns (Figs. 15 and 16, Plate II). These columns also rest in ribs cast on the foundation and mantel-pieces, but are entirely independent of the furnace plates, so that any of these may be taken away without diffi-

* The figures and letters in all of the details of these furnaces are the same on all the plates.

culty. The foundation-plate rests on the ground, and has no masonry under it. The foundation and mantel pieces are alike, except that the foundation piece is open on the cinder side, and has consequently only three sides, while the mantel-piece has all four. Between them all and resting on the bottom is the bottom plate (Fig. 11), which is water-jacketed, and has bevelled edges, a section and elevation of which are shown. Fig. 14 shows the under side of the mantel-piece with a section through *G G*, which rests upon the front and back columns (Figs. 15 and 16), and which is used to support the brick stack above, which carries the hot-air pipes. Immediately above the mantel-piece are two guard-plates, *a a*, (Figs. 1 and 3, Plate I), whose object is to protect the hot-air trunk or bed pipes (Fig. 12, Plate II; Figs. 1, 3, and 4, Plate I), whose supports are shown at Fig. 12. A section at *A A* (Fig. 1, Plate I) is seen in Fig. 5, Plate II, showing the L-shaped corner pieces, Fig. 8, which support the brick stack containing the hot-air apparatus. These corner pieces are made of cast iron, and are held at the bottom by projections cast on the upper side of the mantel-piece, and at the top, Fig. 6, by similar projections on the bottom of the top plate. The binders (Figs. 2 and 7), which tie the corner pieces together, are also made of cast iron, and rest on little projections cast on the corner pieces.

Figs. 1, 2, 3, and 4, Plate I, show respectively a front elevation, a side elevation, a section *B B*, and a half section *C C*, of the furnace with its various appendages. Beside the furnace next the tuyere is a water tank, called a bosh trough, for cooling the tools, through which the water needed during the operation passes. All the plates of the fire-box except the bottom plate are exposed to the air, which with the water-jacketed bottom and repair pieces, adds materially to the life of the furnace.

At Saranac the construction of the five-pipe furnace is slightly different; the tuyere plate, instead of being made of one piece, is cast in two pieces, the upper one of which is called the merritt, and the lower one the tuyere plate, each of which is water-jacketed, and has no repair piece (Figs. 1, 3, and 13 *b*, Plate III). The opening for the tuyere is in the merritt plate. Through the water jacket they are six inches thick, while at Ironville they are only three inches. No box is cast on them to hold the tuyere, as at Ironville, the thickness of the plate being sufficient to insure a tight packing.

The fore-spar plate is also made of two pieces (Fig. 1 and Fig. 13), the lower one of which is called the fore spar, which is set at a

slight angle; the upper one, called the skew plate, is set at the same angle (Fig. 1). The fore spar is somewhat thicker than the skew plate, and has a repair piece with dovetailed edges to keep it in place. It can only be removed by taking out the skew plate above. The hair plate (Fig. 13 *a*, and Fig. 1) rises to only half the height of the back, which is built up in brick. The cinder plate (Fig. 13 *c*) and the fore plate (Fig. 9) are the same except in details. The bottom plate (Fig. 11) has square instead of bevelled edges. The water jacket, as compared with that of the furnace at Ironville, is small. It is nearly square, and is so large that all the side plates of the fire-box rest on it. The foundation of the furnace is made of stone lined with red brick. The side plates are inclosed in brickwork covered with stone. The mantel-piece (Figs. 1 and 14) rests on foundation-pieces (Figs. 3 and 17), and extends only over the front of the furnace. The foundation-plates run at right angles to the tuyere, and support the side walls of the stack. There is no top plate in the furnace. The brick is drawn in and then carried up as a chimney, whose sides are secured by flat bars (Fig. 1), pierced at regular intervals by inch holes through which bars run, which are keyed up at the sides to bind the walls together.

The three-pipe furnace (Plate IV) differs but little from the five-pipe furnace. All the side plates are the same, except that the skew plate (Figs. 1 and 13 *b*) inclines at a greater angle than the fore spar. The other plates are vertical. The fire-box is three inches deeper, but the area is the same. In the five-pipe furnace, the height to the top of the merritt plate is thirty inches, while in this furnace it is thirty-eight and a half. The bottom plate also differs slightly, as shown below:

	Length.	Width.
Five pipe, . .	32 inches.	30 inches.
Three pipe, . .	39 "	42 "

The size of the water jacket is also different.

	Length.	Width.	Thickness.
Five pipe, . .	18 inches.	16 inches.	5 inches.
Three pipe, . .	42 "	14 "	6.5 "

The side plates all rest on the bottom plate as in the five-pipe forge. In other respects the furnaces are alike.

The following table gives the weights of each of these plates :

	Ironville.		Au Sable Forks.		Saranac.	
	No.	Lbs.	No.	Lbs.	3 pipes Lbs.	5 pipes. Lbs.
Cinder plate........................	1	300	1	535	300	215
Merritt plate Tuyere plate }	1	900	1	850	450 400	636 654
Repair pieces for tuyere plate.....		175		300		
Fore-spar plate Skew plate }	1	900	1	850	350 380	373 545
Repair pieces for fore-spar plate.		150		450		
Fore plate.........	1	250	1	327	450	315
Hair plate.....	1	500		1070	200	218
Bottom plate	1	850	1	1040	700	600
Foundation plates.......................	1	800	4	570	1000	1500
Mantel-piece................................	1	1200	4	2530	500	600
Wind trunks	2	100 150	2	250	200	400
Hot-air pipes..............................	4	3400	3	3225	2400	4000
Pillars......	2 at 300 2 at 250	600 500	4	1020		
Supports for bed pipes...............	2	150		300		
Bed pipes	2	450 500	2	1050		
Corner pieces for stack...............	4	2000	4	1000		
Top plate for corner pieces.........	1	400				
Binders for corner pieces............	6	300	6	300		
Guard-plates in front of fire......	2	500	2	400		
Iron trough	1	200	1	225		

The weights differ a little in different forges using the same patterns and even in the same forge; as the pieces are recast to replace those worn out their shapes are slightly modified. The total weight of the castings at Au Sable Forks is now 14,452 pounds.

The engine is generally a simple, not to say a rude one run by water, of which there is usually more than sufficient. But as water-power is uncertain they are sometimes obliged to stop for want of power.

At Au Sable Forks a 56-inch turbine runs a Kingsland bellows consisting of three horizontal oscillating cylinders, 60 inches in diameter, and a 60-inch stroke, making 11 revolutions per minute. The piston-rod was furnished, but the rest of the engine and the fitting of the piston was done for $3500.

It was built for an eight-fire forge and made 12 revolutions, furnishing 346 cubic feet of air per minute at a pressure of $1\frac{1}{2}$ to $1\frac{3}{4}$ pounds. It is used for only four fires.

The blast mains and the trunks leading from them are usually of sheet tin, and are supported on the walls of the building so as to be easily accessible. Every furnace has a special conduit branching from the main trunk, and a valve for cutting off the supply of air so as to be entirely independent of the other furnaces. A pressure gauge is usually placed on the main near the forge. The pressure of the

blast is generally not over two and a half pounds, and it usually varies between one and a half and two and a half; at Au Sable Forks it is $1\frac{3}{4}$ pounds. It is regulated by the kind of iron which is to be made. The greater the pressure the greater the yield of iron, but the poorer the quality. The blast in these furnaces is always heated.

Above the walls of the furnace, resting on pillars which support a mantel-piece (without any brickwork underneath as at Ironville, Plate I, or with brickwork around the furnace as at Saranac and Au Sable Forks, Plates III and IV), is a brick stack, which narrows gradually toward the top, and is intended to receive the hot-air pipes. At Ironville it is braced with angle irons at the corners, and tied together with rods or binders, which are fitted to and fastened to projections in the angle irons (Plate I, Fig. 2), which are also used for the purpose of separating the brick walls into sections, so that any part may be removed independently of the rest, when it is necessary for purposes of repair to take out any part of the brickwork.

The shape of the stack is either square or rectangular, depending on the number of air pipes. It is usually one brick thick. Its size is sufficient to easily receive the whole furnace under it. The stack and chimney together are eighteen to twenty feet high.

The pipes are of the old Calder pattern, with knobs cast on them, to keep them apart from each other and the walls of the stack, thus insuring the free circulation of the hot air. The pipes rest on trunks which are fitted to receive from three to five pipes. These pipes are placed on opposite sides of the stack and at right angles to the direction of the tuyeres. The bed pipes are usually protected against the direct action of the flame of the furnace by guard pieces placed in front of them, as shown in Fig. 1, Plate I. The pipes are generally elliptical, but the old circular and least advantageous shape is sometimes retained. There is a great variety of opinion as to the number to be used; at Saranac there are three and five; at Au Sable Forks, three; at Ironville, four; at Russia, five, but three or four are the usual number. The experiment of using horizontal pipes of the Wasseralfingen pattern was made, but does not seem to have been successful, probably on account of the greater number of pieces and the complicated character of the joints to be kept tight. The air is heated to from 600° to 800° Fahr. It now rarely falls as low as 600°, though formerly the attempt was made to run these furnaces with cold blast.

The hotter the air the less fuel will be used, and the greater the

product, but the more likely will the impurities go into the iron. Any decrease in the fuel or increase in the make is dearly purchased at the expense of the quality of the iron.

The tuyere is either D-shaped, laid on the flat side, or oval. It is made of wrought-iron plates three-eighths of an inch thick. At Saranac they are twelve inches long and four and a half inches wide, by three inches high on the inside, and seven and a half by three and three-quarter inches at the outside end.

The nozzle is ten inches long on the inside, and is one and three-quarters inches wide by seven-eighths of an inch in the clear. It is bevelled, the top projecting into the fire-box one and a quarter to four inches beyond the belly. The construction of these tuyeres is difficult and they can only be made by an expert workman.

The inner and outer shell are carefully bent over mandrils of cast iron and the edges welded. The collars are then welded to the inside shell. The outside shell is fitted over the inside and welded to the collars. It is so difficult to make them that the blacksmiths who can do it get $9 each for them.

One tuyere only is used. This was formerly placed to one side of the centre, but the practice seems now to be general to place it in the centre of the tuyere-plate side of the fire-box. The attempt was made to have as many as five small ones, but it does not appear to have been successful.

When the tuyere is fitted to its place the rest of the opening in the plate is filled by a saddle of cast iron, which fits over it and is wedged to its place, and luted with clay. The blast main is 16 to 18 inches in diameter. The blast conduit or trunk is 6 to 8 inches in diameter, and is attached to the nozzle by means of a sheet-iron boot (Fig. 1, Plates I, III, and IV), which fits a little loose on it so as to be made to slide up and down easily, and is secured at any height by a wrought-iron hoop, which can be tightened by two screw bolts.

At Crown Point the tuyere opening is surrounded by a box eight inches square, cast on the outside of the tuyere plate, to hold clay plugging, which makes the tuyere tight (Figs. 2 and 4, Plate I). The height of the tuyere above the bottom of the fire-box is generally between eleven and fourteen inches. It was formerly made sometimes as low as eight inches, but this is not often done now. In itself it is not of very great importance. Raising it is generally equivalent to an increase, and lowering it to a decrease in the make or yield. It must, however, be kept within certain limits, for if it is too high

the "coal crust" accumulates on the under side of the loupe, and this raises the bottom of the furnace and increases the waste of ore.

The angle of the tuyere is, however, very important. It does not vary very much from 14°, though there is no rule for setting it, and every man sets his own, and has reasons which are conclusive to him for wishing it set at one angle rather than another. If the angle is too low and the tuyere lies flat, the bottom of the furnace rises from the accumulation of coal crust, and its capacity is diminished. If it is too high and the tuyere is too much inclined, the blast will cut through the loupe. A high angle is economical in fuel but gives a poor yield in iron. Too flat an angle gives a greater yield, but does so at the expense of the fuel. The blast should strike the loupe with a slight inclination in order not to cut it. The shape which the loupe takes is the best guide to the angle of the tuyere. If it is narrow and the rim very deep, the iron will be hard and steely, and the tuyere is too much inclined. The angle is lessened until the loupe of the proper shape is obtained. The diameter of the nozzle affects the working of the furnace; if it is too large the pressure is diminished, and the make is lessened; if too small the pressure is increased and the impurities are likely to go into the iron. The pressure of the blast must be taken into account also. If the pressure is great so must also the angle be, and *vice versâ*. The projection of the tuyere is usually about four inches. The tuyeres were formerly reset frequently, but now, when once set, they remain in position until removed for repairs, which may be many months.

The cost of constructing such a furnace at Au Sable Forks is estimated to be:

4000 brick,	$40 00
Lime and construction of stack,	25 00
Four long bolts,	12 00
Fitting plates,	4 00
Bolts, plates, gripes, etc.,	14 00
Putting in fire pillars and chimney, plates, etc.,	10 00
	$105 00

The total cost of the furnace will be between $550 and $600. The object of the process being the production of wrought iron directly from the ore, the first thing to be done is to reduce the ore in contact with carbon, and then to weld together the spongy mass of metallic iron in the bottom of the furnace into a more or less solid mass or loupe. To perform the operation of reduction only, a high

temperature is not necessary, but as the reduced iron is in grains of the form of the fine ore, the temperature of low welding heat is necessary to bind the particles together. Any temperature higher than this is likely to introduce impurities, notably carbon, which will make the iron hard and steely, or may even produce cast iron. This is the reason why soft-wood charcoal is so much preferable in this process than that made from denser woods, which would give too high a temperature. It is not easy even with the best conditions to get a soft iron. It is almost always steely and is used both on account of its purity and because it is not entirely homogeneous, in the manufacture of open-hearth and of cast steel.

The silica, which is almost the only impurity in the ore, unites with the oxide of iron and forms a fusible slag which covers the face of the loupe, and by preventing the direct contact of the blast and of the carbon, prevents both oxidation from the blast and carburation from the fuel of the iron in the loupe, and assists in keeping the particles of reduced ore free to weld together. If there were no silica in the ore it would have to be added. It is an absolute necessity that some of the ore should be sacrificed to protect the iron. As the temperature is low a considerable portion of the sulphur and phosphorus are eliminated and go into the slag. The temperature during the operation is not sufficiently high to again reduce the phosphorus which has passed into the slag. It would not do, however, to use ores high in phosphorus, as beyond a very small percentage there would be no certainty of its being slagged off.

When a furnace is cold two baskets of charcoal dust are thrown into the furnace to protect the bottom plate; it is then filled with large charcoal and lighted, and the blast turned on little by little. It takes several hours to bring a cold furnace up to the proper temperature, but when an operation has just been finished a new one is commenced at once by turning on the blast and loosening up and stirring round all the material in the fire-box. The cinder separated in the process of shingling a former loupe is thrown in, two baskets of charcoal added and heaped up, and the ore charged.

The charcoal baskets (Fig. 16, Plate V) which are used are made of ash, and are deeper on one side than on the other. This deep side is called the belly. The shallow side is turned to the fire so as to empty all the charcoal out; and is usually protected with a covering of sheet iron 4 inches deep. To make these baskets pieces of ash wood of the desired width are beaten with a mallet to make them split along the lines of graining.

The charcoal is brought from the kilns and stored in the charcoal yard. The baskets are filled there as full as they can be and not have the coal fall off, and are piled on the charcoal car (Fig. 15, Plate V) as high as a man can reach. The cart will hold about two dozen baskets. One man brings the cart into the furnace house, and piles up the baskets in a place convenient for the bloomsmen. In the charcoal yard there are always at least two men picking up the charcoal and separating the screenings. The large screenings are always saved, and each bloomsman is expected to use at least one basket of them and sometimes two in a heat. They are generally added somewhere about the middle of the operation. Each basket holds from one and a half to two bushels, the quantity depending on the way it is heaped. The size of the charcoal used must be kept within limits. If it is too large the ore, having too little obstruction, slips through and is likely to reach the loupe without reduction. If it is too small the reduced iron is prevented from reaching the loupe and is likely to become carburized, and the fire is likely to become clogged so that there is not sufficient heat.

A certain amount of cinder is always charged at the beginning of a new operation, if it has not been left in the furnace from a previous one, the quantity required being from one to three shovelfuls. The ore is brought to the furnace in carts. Several tons of it are piled up in a wooden bin to the right of the furnace. This bin is closed on three sides, but is open on the side towards the furnace, and is within easy reach of the bloomsman.

The ore-shovel used is the ordinary long-handled one (Fig. 13, Plate V). A shovelful of ore weighs about thirty-five pounds. At Saranac this weight is often added, but at Au Sable Forks not more than half this weight, and sometimes much less, is put on the fire at a time. The ore is charged as soon as the temperature is high enough. This is done by a peculiar shaking motion of the hand, which distributes it evenly over the fire. The loupe commences to form in from ten to fifteen minutes. Both the quality and the quantity of the iron, as well as the amount of fuel used, depend on the temperature being kept even throughout the process.

To ascertain what is going on the workman watches the color of the flame, and the color and fluidity of the cinder. He also sounds the furnace every few minutes with his furgen or tempering bar, which is an iron rod five feet long and one inch in diameter, with one end slightly rounded, and the other terminated by a knob which

serves as a handle (Fig. 1, Plate V). This is done to ascertain the way in which the furnace is working, to determine the shape of the loupe and how it lies in the fire-box, and to a certain extent its quality, by the appearance of the cinder or of the iron button which attaches itself to the end of the furgen, according as he probes the slag or the loupe. To keep an even temperature the charcoal must be charged regularly, but to have the reduction work successfully the ore must be charged irregularly, both as to time and quantity; the object in charging the ore being twofold, to have sufficient ore to form the loupe properly and to keep the temperature down to the point where the maximum effect will be obtained. When the charcoal flames too much or the heat of the fire is so great that it is difficult to work at the furnace, the bloomsman throws water from the bosh trough (which he dips out with a piggin) over the fire and the front of the furnace. The water cools down the surface of the fire and thus concentrates the heat, keeping the scoria melted. It also prevents some of the fine ore from being carried off, and in this way diminishes the quantity of "emery" that forms.

The charges vary from one to three shovelfuls at once, depending on the way the furnace is working. The tables made at Saranac and Au Sable Forks (page 534) show that the quantity of ore charged is quite different in different works. At Au Sable Forks the amount of ore added at any one time is very small, but the intervals are very short, while at Saranac the quantity is larger and the intervals longer.

The flame, when the operation is being properly conducted, will be bluish or reddish. When it changes to brilliant white, tipped with yellow, it is said to "flash," and shows that the temperature is too high. The fire must then be chilled at once either by charging ore, the quantity depending on the brilliancy of thê flame; or, sometimes where the charcoal has burned low, by taking off some of the fuel. The charge above the tuyere thus becomes very light and the blast comes up through, or "blows out," throwing the coals to one side, and allowing the cinder on the face of the loupe to be exposed for a few minutes. This blowing out sometimes happens before the coal is removed. The flashing should not be allowed to occur frequently, and some workmen do not allow it to happen at all. It generally takes place from inattention on the part of the workmen, though it is sometimes done intentionally to cool the face. There is always danger whenever the temperature becomes too high,

even for a very short time, that some of the iron will become steely, and the loupe will be less homogeneous. To avoid it the fire-box should be constantly sounded, and the heat of the fire regulated by the indications of the furgen, which must be thrust into different parts of the face of the loupe, and moved round its sides to ascertain its shape, the condition of the rim, and how it lies in the fire-box, and plunged down into the scoria to ascertain both its quality and quantity. The bloomsman is thus guided in his work by the hardness of the loupe, its shape, and the way it lies in the fire-box, the color and fluidity of the cinder, and the color of the flame. If the loupe feels hard, if the cinder flows sluggishly and is of a red color, the temperature is too low, and additional fuel is necessary. As the slag is not always ready to be drawn from the tap-holes the condition of the cinder is ascertained by plunging the furgen into it and drawing it quickly out. The slag which adheres to the tool gives the necessary indications. After looking at the color it is immediately plunged into the bosh trough and the slag removed by tapping it with a small hammer. When the slag attached to the furgen is of the right temper it is thin and sparkling; when too cold it is thick and white; when too hot it is red. When the fire has been left for some time and there is too little cinder the iron becomes dry and crumbly, and hammer scales are added. A hot cinder turns red, scintillates, and is very uneven on the surface when cold. When there is too much cinder the loupe is soft.

The time to tap the slag is judged of by a peculiar noise of the blast, which is called fluttering. This usually occurs when it is four to five inches deep. The tapping is done with the cinder bar, five feet long and one inch square, pointed at the end, with which the holes in the cinder plate are pierced. When this bar gets bent it is straightened with the small hammer on an anvil, which stands at the right of the bloomsman, near the ore bin. The slag should be a protoxide slag, which may be easily tapped. When it becomes pasty, fine ore is thrown into it, which dissolves without being reduced. The proper temperature is indicated by a pasty condition of the loupe, and an easily flowing cinder. It is very easy to distinguish between the proper temperature and a too hot working. When the loupe is sounded with the furgen, if the temperature is right, the tool will sink but a short distance into the face of the loupe, and when withdrawn but little of the iron will stick, and there will consequently be only a short thin thimble of iron with a round knob at the end called a button attached to the end of the furgen. If

the temperature is too high the loupe will be soft, the tool will sink deep, and a long thick thimble and button will be drawn out, in which case ore must be charged at once. Before doing this the fire must be "tightened," which is done by pushing the stock towards the tuyere and pressing it down with the fire or ore shovel (Figs. 13 and 14, Plate V). If this was not done there might be danger of the ore being charged on the bare face of the loupe, and thus chilling it. It is also best to tighten the fire before adding fresh fuel. When the loupe has a slippery or pasty feeling, and the cinder is fluid, and the button at the end of the furgen is small, the furnace is working well, and the heat should be kept up to this as the working temperature. If the furnace is too cold the furgen will not enter the loupe, but will strike against it with a dull thud; no attachment will take place. In this case charcoal must be added, and the furnace left to itself until it becomes hot enough to work. When for any reason it is necessary to raise the temperature by adding a large quantity of charcoal, the fore bar is placed on the fore plate, and the charcoal piled up against it. The fore bar is generally used in the first part of the operation, just after the reheating, but is always removed before the close of the operation. The loupe should commence to form at the centre of the fire-box. It should always be saucer-shaped with a hollow in the centre called the face. It should have a rim around it, which must remain unbroken during the whole operation, so that the face of the loupe may always be protected by slag. This rim, during the first part of the operation, is about one inch thick; towards the close it becomes thinner on the tuyere side, and much thicker on the fore-spar side. If for any cause the rim is cut or broken the slag runs off from the face, and the iron is in danger of both of oxidation and carburation; and this will be likely to take place if the loupe is not in the centre of the fire-box. So long as the rim is intact, the loupe is said to have a good face. When it becomes broken it must be repaired at once by charging ore directly over the broken spot, and by working the sides of the loupe with the furgen. Such repairs are very difficult to make, and can only be done very slowly. As the repairs always take time, during which the loupe is without a cover and is exposed to the direct action of the fuel, the iron made from such a repaired loupe is likely to be hard, and rarely makes a good bloom. If the rim is preserved the exact shape of the loupe is not of great consequence so far as the loss of ore is concerned, but when it is badly formed it is difficult to handle it.

This accident of breaking the rim is likely to occur in the course of the operation from three different causes:

1. From an improper position of the tuyere.
2. From starting too cold.
3. From an excess of cinder in the fire-box.

(1.) When the tuyere is out of centre the loupe stretches out and becomes contracted, the face becomes deep, the rim forms unevenly, and finally a gutter forms which allows the slag to run off as the face rises. The same will occur when an obstruction on the nozzle of a correctly placed tuyere forces the blast from its normal direction. The nozzle must therefore be watched to see that it has the right projection and is free from obstructions.

(2.) When the furnace is started too cold, or when the operation has not been properly commenced, as when too much ore has been charged at first so that the fire has become chilled, a large amount of coal crust will be made in the bottom of the fire-box, the loupe will commence to form on it, and will consequently be too high on the bottom of the fire-box. When the temperature subsequently becomes high, a portion of this coal crust will melt, and the loupe will be liable to sink irregularly, so that the face of the loupe will be destroyed. This coal crust contains some metallic iron, but is composed mostly of unreduced ore. It varies in composition with the ore. The only essential change from the ore is a small reduction of the amount of sulphur. As it is below the zone of reduction it is very liable to fuse when the operation commences or before the face of the loupe has become thick. The following are two analyses of it:

ANALYSES OF COAL CRUST.

	No. 1.	No. 2.
Metallic iron		14.16
Protoxide of iron	35.52	53.82
Sesquioxide of iron	43.98	
Silica	12.05	21.07
Titanic acid		3.19
Manganese	0.19	0.37
Alumina	2.30	1.99
Lime	3.71	3.36
Magnesia	1.64	2.07
Phosphoric acid	0.30	0.09
Sulphur	0.04	0.12
Carbon		0.12
Water	0.28	
	100.01	100.36

No. 1. Saranac (Hasegawa). No. 2. Au Sable Forks.

It is always safest to start the loupe moderately hot, rather than

cold, because the "hot back" is stronger than the cold. With a cold bottom the metal will be liable to liquate, and will drop on the bottom and chill. The fire must then be worked hot, but the loupe will show white, hard spots, where the liquation has taken place. If the back has been too cold, it may happen that, from the very irregular way in which the face of the loupe sinks, it will be impossible to repair it. The loupe formed will be worthless, and will have to be dug up and the operation begun over again. Such a loupe will have to be cut up hot, and added to the furnace little by little in several operations, or used to make an impure quality of iron. It would not be worth while to make a billet of it.

(3.) If the cinder is allowed to accumulate in the fire-box the loupe will be thin and the rim flat, instead of thick with a deep rim, as it should be; the temperature on the face of the loupe will be high, and, as it is not protected from the fuel, will be likely to become so highly carburetted as to melt the rim. At such a temperature the cinder is likely also to cut the rim, and there is thus a double danger. Besides this, when the temperature is high, the loupe is also liable to be more impure. The danger of carburation makes it necessary to have some cinder in the fire-box in order to protect the loupe. It should be tapped from time to time, generally about once an hour, so as not to be allowed to accumulate beyond what is necessary. The workman will have to judge between the danger of breaking the rim and of making hard iron, and will regulate the tapping of the cinder accordingly. It may happen that, when there is not enough cinder, he may be obliged to add some from a previous operation, but this is not usual. The slags which flow from the different tappings do not differ essentially from one another. The first slag drawn while the loupe was just forming contained 11.6 per cent. of metallic iron, which was carefully separated with a magnet before the analysis was made, and does not appear there; the second only 3 per cent.; the last none at all. No. 3 was from the bottom of the loupe just before digging up.

This slag runs into the cavity under the fore plate, where, when it does not cool rapidly enough, water is thrown upon it. It is then shovelled into some convenient place, and when sufficient has accumulated it is loaded in an iron wheelbarrow and thrown away. In cleaning it up from the forge floor (and to serve the general purposes of a broom), the scraper (Fig. 17, Plate V) is used.

Just before the loupe is finished the lowest cinder-hole is always opened, but very frequently the cinder will not run from it, while it always does from the one above, away from the tuyere. The slag

from all the tappings is collected in a pile near the furnace and carried to the dump in an iron wheelbarrow (Plate V, Fig. 19).

ANALYSES OF SLAGS.

	No. 1.	No. 2.	No. 3.
Metallic iron	3 19	3 68	1.24
Protoxide of iron	48 57	49.74	49 67
Sesquioxide of iron	8 06	4.93	11.17
Manganese	0.61	0.40	0.64
Alumina	1.60	0.80	
Lime	5.54	5.37	6.16
Magnesia	2.29	2 22	2.29
Titanic acid	1.36	6 26	4.46
Phosphoric acid	0 03	0.40	0.05
Silicic acid	26.38	24.60	25 93
Sulphur	0.25	0.37	0 00
Carbon	1.18	0.33	0.22

It is interesting to note the concentration of the phosphoric and titanic acids in slag No. 2, and a subsequent diminution in No. 3.

Ore is put on up to the last minute if the fire will bear it. During the whole operation the furnace should have just as much ore as the fire will bear and remain at the requisite heat. Towards the close of the operation but little fuel is added. The stock is moved toward the tuyere and pushed up from the front, and everything is got ready to dig up the loupe. It is always most economical in fuel to keep the ore up to the full capacity of the fire ; but to do so requires an active and intelligent bloomsman. There is a very great difference in the capacity of the men in this respect. Some men will always produce a maximum yield of good quality with a minimum consumption of fuel, while others in the same forge will not succeed so well, owing generally to inattention to their work.

There is a great variation in the quantity of ore used in each operation, depending largely on how much was left behind from a previous one.

The following tables give the charges of ore for five operations. Table No. 1 was observed for me by Mr. Hasegawa at Saranac. Table No. 2 was observed by myself at Au Sable Forks.

No. 1.

CHARGES OF ORE IN FOUR HEATS.

	1st heat.		2d heat.		3d heat.		4th heat.	
	Time, P.M.	Ore.	Time, P.M.	Ore.	Time, P.M.	Ore.	Time, P.M.	Ore.
	12.	0	3.	0	6.10	0	3.	4
	12.40	2	3.15	3	6.45	2	3.55	2
	1.	2	4	2	7.20	2	4.05	2
	1.25	2	4.25	2	7.40	2	4.25	2
	2 25	3	4.40	2	7.50	4	4 50	1
	2.40	2	4.50	2	8.45	2	5.05	3
	3.00	2	5.06	3	9.05	3	5 25	3
			6.10	2			5.45	3
							5.55	4
							6.05	3
Length of operation............	3 hours		3.10		2 55		3.05	
Total ore in shovels............		13		16		15		27
Baskets of charcoal..............	20							
Bushels of charcoal	40							

No. 2.

Time.	Ore in shovels.	Charcoal in baskets.
9.25 A.M.		2
9.30 "		1
9.35 "	2	
9.36 "	3	
9 37 "		1
9 47 "	1	1
9.50 "	1	
9 51 "	1	
9.55 "	2	2
10.03 "		1
10.12 "		1
10.20 "	2	
10.21 "	1	
10.25 "	1	
10.26 "	2	2
10.27 "	2	
10.33 "		1
10.38 "	1	1
10.44 "		1
10.52 "	2	1
10.56 "	1	1
11 07 "	1	1
11.15 "	3	1
11.25 "		1
11.28 "		1
11.34 "	2	
11.36 "	2	1
11.43 "		1
11.50 "		1
11.55 "	1	
11.58 "		1
12.02 "	2	1
12.11 "	1	
12.21 "	1	1
2 hours, 56 minutes.	35	26

When the loupe is ready the charcoal is shoved from the front to the back of the fire-box, and the blast is turned off. The signal is then given to the hammerman off duty, who tries the square holes in the cinder plate, and, selecting the one from which he thinks the loupe can be most easily raised, he inserts the wringer (Plate V, Fig. 18) into it, and pries down on the bar, in this way loosening it and freeing it from the sides of the furnace, and turning one side up so that the charcoal falls into the fire-box behind it. Hammerman No. 1 now dampens down the loupe and the fire with water from the piggin. If the loupe is free the hammerman and bloomsmen are able to raise it; but if it is attached to the sides, or caught under a projection of the tuyere, four or five men come from the neighboring fires and pry with a longer bar to raise it. The loupe now lies side up in the fire-box. Hammerman No. 2 then withdraws the wringer from the cinder plate, and, placing it under the loupe in the furnace, pries it up, using the edge of the fore plate, which has been cleaned off, as a fulcrum, while the bloomsmen of this and the neighboring furnace "dig up" the bloom with their foss hooks (Plate V, Fig. 2), by catching it on the rim and drawing it upon the fore plate, where the coal crust is separated, and the loupe cleaned off with the foss hook by hammerman No. 2. During this time hammerman No. 1 again throws water from the piggin on the loupe, to keep down the heat. When the coal crust is separated the loupe cart (Fig. 11, Plate V) is brought by the bloomsman, who, with the assistance of the bloomsman from the next forge, rolls the loupe upon the cart and carries it to the hammer. All the stock that falls from the loupe is put back into the furnace. The men help each other about all this work. There are two hammermen for each hammer, who take turns in their work, each one hammering alternate blooms. The one not hammering is said to be off duty. Generally the hammerman off duty, and the man who tends the next fire, do the work of stirring the fire, loosening up the material in the bottom of the furnace, putting in the charcoal and turning on the blast for the next operation, while the forgeman is at the hammer.

During the process an infusible material, called emery, collects on the guard-plates and in the hot-air chambers. This must be carefully removed from time to time. It is composed, as seen by the analyses given below, of a silicate of the sesquioxide of iron. If not removed, it is likely to fall down into the fire-box at any time. As it

often contains a large percentage of phosphorus, and is very refractory, it will deteriorate the iron.

ANALYSES OF "EMERY."

	1	2
Sesquioxide of iron	64.79	60 21
Protoxide of iron		23.55
Silica	18.35	9 56
Manganese	0.20	0 34
Lime	5 78	1.04
Magnesia	7.03	1.00
Alumina	1.60	2.74
Phosphoric acid	1.00	0.17
Sulphur	trace	0.07
Water	0.26	
Carbon		0.86
	99.01	99.54

1. Saranac (Hasegawa). 2. Au Sable Forks.

The tools required for a forge at Au Sable Forks are (Plate V):

1 Bloom tongs with seven-inch jaws.*

1 Turn-bat of wood two feet six inches long and one and a half inches wide.

1 Billet tongs with three and a half inch jaws.

1 Ore shovel with a handle four feet long.

1 Fire shovel with a handle four feet long.

1 Foss hook with a looped handle, six feet long.

1 Cinder bar half an inch square, five feet long.

1 Tapping bar half an inch in diameter.

1 Furgen.

1 Wringer seven feet long.

1 Sledge weighing three pounds, with handle two feet long.

1 Hammer weighing one and a half pounds with handle one foot long.

1 Fore bar two inches square, and the length of the fore plate.

1 Anvil.

1 Piggin for water.

When the loupe, which weighs from three hundred to four hundred pounds,† is dumped at the hammer front, it is turned over from the cart, with the face down, or up, as may be. It is about twenty-seven by twenty-two inches, and twelve to thirteen inches

* At Saranac there are two sizes of bloom tongs (Figs. 5 and 6, Plate V).

† The one which I weighed from Au Sable Forks, was four hundred and twenty-three pounds.

thick. The bloomsman cuts into it on the longer side with an axe, which is shown in Fig. 3, Plate, V, the ordinary woodman's axe, sometimes single and sometimes double edged; the hammerman and bloomsmen then turn it over with the foss hooks, so that the face side is up. It is then hacked on the rim. The object of this is to make a place into which the loupe grampuses (Fig. 4, Plate V) can fit. Hammerman No. 2 then seizes the loupe with them. The end of the handle of the grampuses turns up so that the hammerman has a firm purchase on them. To lift the loupe on to the anvil block after it has been hacked, the hammerman puts the jaws of the loupe grampus under the loupe and opens them wide; this raises it a little. The bloomsman then supports it with a bar. The hammerman then seizes it, and by an extremely dexterous movement, rolls it over the bar upon the first step of the anvil. On this step the action is repeated and the loupe rolled upon the shingling die under the nose of the hammer. The bloomsman then raises the water-gate of the wheel, and the hammering is commenced. It is divided into three phases:

1. Shingling, or breaking down the loupe.
2. Drawing out.
3. Smoothing.

To perform these operations the anvil block and the hammer head are made of different widths and sizes of chilled cast iron. The end, where the face is very large, is used for shingling, and for this purpose only light and slow blows are required. The middle is used for drawing out, the blows being rapid. The inner side is used for smoothing. The object of the shingling is to bend the rims of the loupe together, leaving the face in the centre, and to hammer them down to a cylindrical or octagonal form, during which time part of the cinder is squeezed out.

As soon as the loupe is on the loupe cart, and before it is brought to the hammer, which up to this time has been resting on the anvil, the hammer is raised to its full height, by partially opening the gate of the water-wheel, and is kept in this position until the bloom is on the shingling die, leaving plenty of room to turn the loupe up on its side on the die, so that the first blows of the hammer will roll the rim over on the face. When the loupe is on the shingling die the water-gate is opened, and the hammering commences. As the bloom is quite soft, a few light blows are sufficient to crush it in. The loupe is held in and turned by the grampuses until it has been thor-

oughly upset, and been brought to the desired shape, which is from seven to ten inches square.

The grampuses are then removed, and the loupe is caught with the bloom tongs (Fig. 5, Plate V). These have an oval ring which fits around both arms of the tongs, and serves as a clamp. To fasten the loupe securely in the tongs, and make the clamp tight, a stick of wood, called a turn-bat, is placed behind the ring, the arms pressed apart, and the ring shoved as near to the end of the handle of the tongs as it can be made to go. The tongs are supported at their hinge in a loop of chain, attached to a crane, which runs over a grooved pulley nine inches in diameter. Above the loop the chain is attached to a screw with a hand wheel twelve inches in diameter, so that the tongs can be placed at any convenient height. The hammerman now strides the tongs and sits on the arms to steady them, resting on them sufficiently to balance the loupe, but bearing on with the whole weight of his body, when he wishes to raise it from the anvil. He seizes with his hands the turn-bat, which rests between the clamp and the end of the tongs, and turns the loupe with it as he needs, pushing the loupe backwards and forwards with his feet until it is hammered to size. The operation is both difficult and dangerous, for the least inequality of force, or want of contact of the bloom with the anvil, would shoot the hammerman up to the roof. The hammerman first hammers a notch all round the middle of the bloom with the smoothing die, hammering it down to nearly the size of the billet. He then hammers the rest from the notch and the ends toward the centre of the half bloom, turning it frequently. The hammer makes seventy-two strokes per minute; eight strokes are given on each side before the loupe is turned over.

During the hammering some cinder is expelled. The nature and quality of this cinder give a very fair idea of the quality of the loupe, but the workmen generally know before the loupe is hammered what kind of iron they have made. If the slag coming from the bloom is liquid, and if the bloom hammers easily, and is without hard spots, it is called of first quality. If the slag is thick, and the loupe hard, and cracks under the hammer, it is of second quality, and may contain at one time iron, steel, and cast iron. When it is to be used in the Siemens-Martin process this is not of much consequence, but if it is to be made into bars, it is unsuitable, as the hard spots make it roll unevenly.

The cinder is not very rich. It contains some metallic iron me-

chanically mixed with it. The analysis of a sample from Au Sable Forks, gave the following result:

Metallic iron,	13 75
Protoxide of iron,	38 31
Manganese oxide,	0.56
Alumina,	0 57
Lime,	7 01
Magnesia,	2 21
Phosphoric acid,	0 10
Silicic acid,	21.62
Titanic acid,	7.68
Sulphur,	0 25
Carbon,	1.10

This cinder, amounting to from three to four shovelfuls for every loupe, is put back into the fire at once. A considerable quantity of hammer scale accumulates around the hammer. This is carefully collected and put in a convenient place near the furnace to be used during the operation. Only half of the bloom is hammered down on the drawing-out die. In this way a four-sided billet, about four by five inches, is made. It is then rolled upon the smoothing die, the irregularities of the surface taken out, and the faces made smooth and even. During this process the corners are flattened. The part so drawn out is generally made into two billets, each of which is seventeen inches long. It is sometimes made into three when the loupes are small, but very rarely. The billet is now brought over the end of the anvil, and the hack (Fig. 8, Plate V) placed on top three inches from the end so that the hammer strikes it. It is cut half through on one side, and is then turned and cut off on the opposite side. The part cut off is called the fag-end; it is so imperfectly welded that it cannot be used, and is taken back to the furnace. Five inches are then cut off to make the billet perfect, which is called the crop end. This is used at Au Sable Forks to make nail plates, and is called the nail piece, or nail chunk. The name of the works is now stamped in with the branding iron (Fig. 9) on the place corresponding to the middle of two billets, by giving it a gentle blow with the hammer. A billet about seventeen inches long is now cut off with the hack, leaving a billet attached to the bloom-head, which is sufficiently long to be seized by the billet tongs. The tongs are then drawn away from the hammer, and are supported in a hook suspended by a chain from a grooved iron wheel running on an iron rod fastened to the top of the crane. This hook is rectangular in shape, and is twenty-four inches long and ten inches

wide, with a vertical turn two inches high on the end to prevent the bloom from slipping out. The billet end is seized with the billet tongs (Fig. 7), and the bloom tongs are removed. Supported on the hook, the bloom-head is plunged into the furnace in which it was made, which during the hammering has been heaped high with charcoal and is well burning, the tongs are removed, and the bloom-head left there to be reheated. When it is sufficiently hot it is hammered as before. To do this the hammer tongs are taken to the furnace, put on the billet end of the loupe-head, which is then lifted out of the fire by the forge crane, supported on the L piece. The handle of the tongs is run through the loop of the wheel chain, the L pieces shoved to one side, and the bloom-head drawn out as before. The loupes from all the furnaces, as well as the reheated bloom-heads, are hammered each in succession in the same way. After each operation of hammering the anvil and the hammer head are cooled with water. After all the loupes are out the L piece of the crane is hung up. The billets weigh from seventy to eighty pounds each; four are made from a loupe. The fag ends weigh eight to ten pounds each; the crop ends, thirty to forty pounds each. The loss in weight from the slag is not large. The reduction in weight from bloom to billet is about five per cent. The billets are thrown on the billet cart (Fig. 12) with the billet grampus (Fig. 10, Plate V), and carried outside the works.

The crop-ends and billets are weighed together, and the workmen paid full price for them. The irons from the different forges are, however, kept separate, as the men are paid by the ton. Each bloomsman takes his iron to the weighmaster on the billet cart. It takes about two to two and a quarter hours to hammer the loupes and loupe-heads to billets from six fires, with the wooden hammer used at Ironville. At Saranac and Au Sable Forks the iron hammers are so much heavier that they require one hour and three-quarters only. After this, as the work of hammering is very fatiguing while it lasts, the hammermen rest until the next bloom is dug up.

The product from a loupe is thus two fag-ends put back into the fire, two crop-ends and four billets. When the bloom does not get a good welding heat all of them are cut up for nail plates. The iron is good and strong, but is full of seams, and on account of these defects of welding, which do not materially weaken the iron, it makes it look badly, so that it cannot be sold as profitably as when made into nails.

During the operation of hammering the blooms a considerable

quantity of hammer scale falls around the hammer, which is carefully collected.

This hammer scale contains a large quantity of shots of metallic iron. This was carefully separated with a magnet and amounted to 27 per cent. The residue contained:

Metallic iron,	22.96
Protoxide of iron,	26.77
Manganese oxide,	0.34
Alumina,	1.35
Lime,	5.62
Magnesia,	2.21
Phosphoric acid,	0.07
Sulphuric acid,	0.18
Silicic acid,	28.20
Titanic acid,	6.38
Carbon,	1.27

This is put to one side to be used in the furnace.

The following table gives the diary of the operation of hammering for four furnaces at Au Sable Forks:

3.03	P.M.	Loupe raised; sprinkled with water to cool it.
3 04	"	Loupe tipped over on loupe cart.
3.05	"	Loupe dumped at hammer, top side up.
3.06	"	Loupe cut and turned over.
3 06½	"	Loupe cut and put under hammer.
3 08	"	Loupe hammered to bloom.
3.09½	"	Tongs on and hammered to billet.
3.34	"	Loupes of all the furnaces hammered.
3.35	"	Hammer tongs put on the first reheated bloom-head and brought to hammer.
3.40	"	Reheating hammering finished.
3.45	"	All the loupes drawn to billets.
3.46	"	Hammer wetted down.
3.49	"	Operation finished.

At 12 o'clock on Saturday night the blast is turned off and the furnace banked up until Sunday night at the same hour, at which time six baskets of coal are sufficient to heat up the furnace. The blast is turned on by degrees.

Instead of billets, slabs and half-blooms are sometimes made. Slabs are loupes shingled down without reheating. They require 250 bushels of charcoal per ton. Half blooms are loupes hammered down to seven or eight inches square after being reheated and cut into two pieces. This requires 275 bushels of charcoal.

Two kinds of hammers are used, the first a light one with wooden

helves, used at the Crown Point Works at Ironville; the other of iron, used at Au Sable Forks and Saranac. Of this latter form there are two types, differing in details of construction only, but interesting in these details.

The wooden hammer is so light that one of them can only serve three fires, while the iron hammers can easily serve four, and can be used for six. The difference in cost is very small considering the greater amount of work done, and the iron ones are consequently the most advantageous. They are built so that all the parts can be very easily transported.

The foundations of the anvil at Ironville (Figs. 1 and 2, Plate VI) are made four feet deep, and four feet four inches square. Timbers ten inches square are laid in this in successive layers, each layer being at right angles to the next, and wedged tightly by driving wooden wedges between them. To bring these blocks up to a proper level for the anvil, planks are laid down. The bottom of the anvil has lugs cast on it, for which places are cut in the timber to prevent its turning or slipping. The anvil block itself is I-shaped, and of circular section, about four feet high, with a hole for the anvil pin. The anvil is fitted and fastened into this (Figs. 3 and 4). The dies, both of the anvil and the hammer head, are dovetailed into their places and fastened there. The smoothing die is square at the bottom, and is set at right angles to the hammer head. The drawing die has the corners cut off, and is at right angles to the smoothing die. The harness is fitted to timbers two feet square which run the whole length of the hammer (Figs. 1 and 2).

Above the hammer is a spring beam of wood (*B*), the object of which is to prevent the hammer going too high on the up stroke, and to give it more force in coming down. The trip-wheel is of iron; where it lifts the hammer a hard wood block is attached so as to prevent wear. The hammer itself is of iron.

The cost of these hammers is given below:

COST OF WOODEN HAMMER AT IRONVILLE.

Foundation and setting up,	$200 00
Framed helves,	25 00
Castings, 1800 pounds at 3 cents,	54 00
Anvil block, 10,000 pounds at 3 cents,	300 00
Stake block, 2000 pounds at 3 cents,	60 00
Harness, 1700 pounds at 3 cents,	51 00
Four dies,	12 00
	$702 00

The iron hammers (Plates VII and VIII) are shown in detail in the drawings. Plate VII is the type used at Au Sable Forks, Plate VIII that used at Saranac. They do not differ in any essential principle, but the Au Sable Forks hammer is more convenient, and has besides two step plates (m and n), which allow of the bloom being more easily brought up on the shingling die. These hammers cost from $1200 to $1500, and are in every way superior to the types used at Ironville. The weight of the different parts is given in the table below:

WEIGHT OF THE HAMMERS.

	1	2	3	4
1 Hammer	11,000 lbs.	10,735 lbs.	11,500 lbs.	1,300 lbs.
1 Anvil block	7,000 "	8 928 "	11,500 "	1,200 "
1 Stake	1,500 "	3,000 "	3,000 "	4,800 "
1 Anvil smoothing die.	115 "	165 "	180 "	125 "
1 Hammer " "	115 "	165 "	180 "	125 "
1 Anvil drawing "	60 "	100 "	125 "	75 "
1 Hammer " "	60 "	100 "	125 "	75 "
Loupe blocks				560 "
1 Collar	1,500 "	1,676 "	1,700 "	1,800 "
4 Tappets	240 "	248 "	300 "	325 "
2 Hammer standards	2,400 "	2,570 "	3,500 "	2,000 "
1 Trip shaft	6,500 "	10,000 "	8,500 "	9,150 "
1 Standard shaft	1,400 "		700 "	
2 Pillar blocks	250 "	700 "	200 "	800 "
2 Water-wheel centres.		3,620 "	5,500 "	

Nos. 1 and 2 are the types used at Au Sable Forks. 3 and 4 those at Saranac.

The strain on the hammer head is very great, and requires a superior iron, so that it alone costs $450. The one now made by the J. & J. Rogers Iron Co., requires 33,952 pounds of cast iron for the hammer, block, dies, stands, saddle, and stake.

The hammers might be run by steam, but they are generally run by water.

At Au Sable Forks the power is an undershot water-wheel 18 feet in diameter, with 4-foot face, making 40 revolutions per minute. Such a wheel requires:

Lumber,	3500 feet.
Bolts and washers,	300 pounds.
Wheel centres,	5500 "
Shaft,	8500 "
Collar,	1700 "
Stand for collar end of shaft,	700 "
Pillar block,	200 "
Bed plate,	3500 "

It costs between $700 and $800.

The following are some analyses of the billets made at Saranac and Au Sable Forks. They show iron of remarkable purity, which, however, is unfortunately not homogeneous.

ANALYSES OF BILLETS.

	1	2	3	4	5	6	7
Sulphur	.008	trace.	trace	.001	trace	trace	trace
Phosphorus	.015	.042	.034	.028	.023	.042	.011
Silicium	.005	.280	.021	.512*	.025	.100	.013
Manganese	.079						
Carbon	.223	.170	.220	.180	.170	.165	.220
Slag			.180	.014	.155	.075	.150

* Probably due to slag.

1 and 6. Saranac (Hasegawa). 2, 3, and 4. Au Sable Forks (J. Blodget Britton). 5. Peru Iron Company (O. Wutn). 7. Chateaugay.

DIARY OF AN OPERATION AT AU SABLE FORKS.

9.25 A.M. Loupe dug up; two baskets charcoal charged; blast turned on; slag discharged and cleaned out; charge loosened up all around the fire-box.

9.30 " One basket charcoal charged; coal heaped up high in furnace; fire sounded; flame bursts out from the three cinder-holes furthest from the tuyere, and from the square hole from which the loupe is raised.

9.32 " The bloom-head, with tongs attached, brought by the crane and plunged into the centre of the fire-box with a slight inclination downward; fire sounded; tempering bar thrown into the trough.

9.35 " Bloom adjusted so as to get coal under it; two shovels of ore charged.

9.36 " Water thrown into the slag-pit and on the fire; fire sounded; bloom adjusted; three shovelfuls of ore thrown on the fire.

9.37 " One basket of charcoal; fire-box sounded with furgen on the tuyere side.

9.40 " Bloom turned round; fire packed down with ore shovel; forge filled with charcoal; wetted down with the piggin; slag taken out from the slag discharge; flame bursts from the three slag-holes.

9.45 " Bloom adjusted; water thrown on fire; furnace sounded on both sides.

9.47 " Bloom adjusted; charcoal pressed down; one basket charcoal; one shovel of ore.

9.48 " Fire sounded.

9.49 " Bloom adjusted; water thrown on; fire sounded.

9.50 " Bloom and fire adjusted; one shovel of ore.

9.51 " Bloom turned round; one shovel of ore; fire adjusted.

9.55 " Bloom tongs taken off; fire dampened; bloom taken to hammer; fire adjusted; two shovels of ore and two baskets charcoal.

10.01 A.M. Fire flashing on tuyere side; fag-end brought back and thrown into furnace; fire adjusted.
10 03 " Bloom-head brought back and plunged into furnace; one basket charcoal.
10.05 " Water thrown on; fire adjusted.
10.07 " Bloom adjusted; fire packed down; coal pushed off the fore plate.
10.08 " Tap-holes all dark; centre one opened; slag flows in small stream; three holes from left to right flame at once.
10.10 " Bloom turned over; fire adjusted.
10 12 " Bloom adjusted; one basket charcoal.
10.15 " Bloom turned round; fire sounded on the tuyere side.
10.17 " Bloom adjusted; fire packed down with ore shovel.
10.20 " Two shovels ore; water thrown on; fire sounded on tuyere side; bloom and fire adjusted.
10 21 " One shovel ore.
10.25 " Bloom-head turned round; one shovel ore.
10.26 " Bloom taken out; fire dampened and sounded; two shovels ore; the fore bar, two-inches square, is placed on the fore plate; two baskets charcoal.
10.27 " Two shovels ore; slag removed; small, fine hammer-scales put in.
10.33 " Fire adjusted and sounded; all the slag openings on tuyere side flaming; one basket charcoal screenings.
10.38 " Fire adjusted; one shovel of ore; one basket of coal; fire sounded; all the slag openings on the fore-spar side flaming.
10.44 " Fire dampened and packed down; one basket charcoal; fire sounded.
10.48 " Fire wetted; sounded; a short iron button brought up with the furgen; slag hole opened.
10.52 " Two shovels of ore and one basket charcoal.
10 58 " One shovel of ore; fire wetted and sounded; short button brought up; one basket charcoal; middle slag-hole flaming quietly.
11.00 " Fire flashes; wetted down and sounded.
11.07 " Slag-hole opened; slag flows freely; fire adjusted; one shovel ore; one basket charcoal.
11.15 " One shovel ore; fire adjusted and sounded; button brought up; two shovels ore; one basket charcoal.
11.25 " One basket charcoal.
11.28 " Fire wetted down, sounded, and adjusted; one basket charcoal.
11.30 " Fire crowded in with the fire shovel.
11.34 " Two shovels ore; fire wetted and sounded.
11.36 " Fire adjusted; one shovel of ore; one basket charcoal.
11.43 " One basket charcoal; fire adjusted, wetted down, and sounded.
11.45 " Slag tapped.
11.50 " One basket charcoal.
11.55 " One shovel of ore; fire sounded; button brought up; charcoal pressed up with a shovel.
11.58 " Fire wetted down; sounded; one basket charcoal.
12 02 " Two shovels ore; fire adjusted; one basket charcoal.
12.03 " Fire adjusted.
12.11 " One shovel of ore; fire adjusted and wetted down.
12.15 " Fire flashes; slag taken out.
12.18 " Fire adjusted.

12.21 A.M. Fire adjusted: one shovel ore; small basket charcoal; fire wetted down and sounded.
12.24 " Lower slag-hole opened with sledge; upper slag hole opened; cinder flows from it more freely than from hole below.
12.27 " Loupe dug up.
12.34 " Bloom hammered to seven inches square.

One five-ton hammer does the work of four to six furnaces.

The tools required for hammering are: 2 pair grampuses, jaws 1 foot 7 inches long; 2 sets hammer tongs, 7 x 3 inches; 2 hacks for cutting; 1 grampus, 3 feet long, for fag and crop ends; 1 turn-bat of wood, 30 inches long, $1\frac{1}{2}$ inches wide.

For the working of the hammer men: No. 1 commences at 12 o'clock Sunday night, and goes off at 10.30 A.M. Monday. No. 2 goes on at 3 A.M. Monday, and goes off at 3 P.M. No. 3 goes on at 12 noon, Monday, and goes off at 10.30 P.M.

A day shift in these furnaces commences at noon and lasts till midnight; a night shift continues from midnight to noon. The "make" of a shift or the product of the furnaces depends on the pressure of the blast, but this refers to quantity and not to quality. It also depends on the reheating, which is done during the first part of the operation. The time for a complete operation is three hours at Au Sable Forks and Saranac. The formation of the loupe itself does not require so much time; the time is consumed in the reheating of the bloom. To make ordinary billets a single reheating only is necessary, which consumes but half an hour. To make very small billets two of these reheatings, which last about an hour and a half, are necessary, during which time less ore is charged than when the fire-box is free. When the loupe is made into four billets the operation is not much retarded, but when there are to be small billets the product or make of a furnace may be very much reduced. With the usual pressure of blast, which is about one and three-quarter pounds, when good iron is required, and four billets only are made, from four to five loupes, weighing three hundred to three hundred and fifty pounds, are made in a shift. At Saranac six loupes of three hundred and forty pounds have been made in a shift of twelve hours, but this was done at the expense of the quality. For blooms of three hundred and fifty to four hundred pounds three hours is the usual time allowed. The bloom is dug up by the clock, and it is the bloomsman's business to be ready when the time is up.

The make of these furnaces is generally estimated as 2600 pounds of blooms, or on an average one ton of billets in 24 hours, or 300

tons a year, a year being counted as 300 working days. It might be considerably increased if the reheating was done in a compartment made for the purpose or in a separate furnace. During all the time the bloom-head is in the furnace but little ore can be added and a great deal of fuel is consumed. The use of the waste heat, by adding a reheating chamber to a single furnace, does not seem to meet with favor. The heat is either too low and there is too great danger of oxidation and consequent loss of iron when the bloom head is not surrounded by charcoal, or the amount of charcoal consumed must be considerably increased. In order to get sufficient heat two furnaces have to discharge their heat into one set of hot-air pipes. Placing two furnaces together, while it is economical so far as the construction is concerned, has been found to be attended with so many disadvantages on account of the difficulty of getting at the various parts of the furnace, that it has been very generally abandoned. The use of a separate reheating furnace, though undoubtedly practicable, does not seem to have been successful. The use of a steam hammer would be a great improvement, were it not that the water is generally free, and the number of forges in the same building is not usually sufficiently large to keep a steam hammer in constant motion. There does not seem to be any way to essentially improve this costly process in these respects, for the moment the conditions under which the improvement in it can be made are fulfilled, some other process becomes at once more economical.

The usual amount of fuel consumed is 300 to 350 bushels of charcoal to the ton. When everything has been weighed and watched, and the pride of the workmen has been aroused, 240 bushels only have been used for the ton. The usual expense is, however, about 300 bushels, generally a little more. Two tons of dressed ore, containing about 65 per cent. of metallic iron, are used to the ton. The iron lost goes to form the cinder. In some cases three tons of ore give two tons of iron. The cinder, as shown by the analyses, is very rich in iron. It is partially used in the process and, but for the concentration of the phosphorus, it might all be calcined and treated as an ore. It does not pay to use it, and as it cannot be transported in most localities to be used in a blast furnace it is generally thrown away.

One bloomsman has charge of a furnace and works a whole shift. He may or may not have a helper, or an apprentice. If he has an apprentice he aids him during the operation, but the bloomsman alone is responsible for the work. Apprentices are rare, as there are

usually many skilled bloomsmen who work at other trades who can be had in case of need, and there is but little inducement to learn the process in ordinary times. The bloomsman is paid according to the grade of iron he makes. Three of these grades are recognized. He is paid $4.75 per ton for No. 1; $3.50 for No. 2, and $3 for No 3. The quality is ascertained by bending and breaking a sample, or simply by its behavior under the hammer.

The prices paid do not represent actual outlay, as the store system is adopted everywhere, and the workmen are paid in orders on the store for a certain amount of goods. In very many of these forges the process itself does not pay, but the store makes a profit, and this makes it possible to run the works, and keep the mining and smelting industry of the country alive.

The quality is generally equal to that of the best Swedish iron. The tensile strength varies. It has been found in many tests to be from 55,017 to 72,000, and, exceptionally, even as high as 89,582 pounds to the square inch. It is not, however, uniform in texture, and is, therefore, mostly used on account of its freedom from sulphur and phosphorus in the manufacture of crucible or open-hearth steel. It is sometimes used for hoes in the place of steely iron. It also furnishes the material for horseshoe nails to several factories.

The following tables give the estimated cost per ton of 2000 pounds of the three grades of iron made. Most of the works prefer to make blooms whose commercial value is about $40. The cost, as given below, will vary slightly in details in different localities, especially as to general expenses, but the materials and labor are about the same in all the works.

BILLETS.

Charcoal, 300 bushels, at 5.5 cents,	$16 50
Two tons of dressed ore, at $7,*	14 00
Bloomsman,	4 75
Hammerman,	2 37
Stockman,	60
Other labor,	90
Repairs, general expenses, interest, and sinking fund,	2 50
	$41 62

* At Au Sable Forks the ore costs $8 per ton and the charcoal 7 cents per bushel.

BLOOMS.

Charcoal, 275 bushels, at 5.5 cents,	$15 13
Two tons of dressed ore at $7,	14 00
Bloomsman,	3 50
Hammerman,	1 87
Stockman,	60
Other labor,	90
Repairs, general expenses, interest, and sinking fund,	2 50
	$38 50

SLABS.

Charcoal, 250 bushels, at 5.5 cents,	$13 75
Two tons of separated ore,	14 00
Bloomsman,	2 50
Hammerman,	1 62
Stockman,	60
Other labor,	90
Repairs, general expenses, interest, and sinking fund,	2 50
	$35 87

As a considerable quantity of ore is likely to fall through the fire and not be reduced, Mr. E. Peckham, in 1871, patented a furnace, in which he proposed, in the first place, to get rid of the emery by making a pocket in the horizontal flue to catch it; to mix the ore with an equal volume of fine charcoal, and submit this in a series of retorts to the waste heat of the furnace, and to reduce it after thirty-six hours in the retorts. This ore is pushed through openings in the front of the retorts, through doors which close by their own weight, into the chamber containing the hot-air pipes, over the furnace on to a shelf, from which it is discharged into the fire-box by means of a door placed directly over the front of the fire-box. The workman takes this ore from the shelf with a small shovel and scatters it over the fire exactly as in the ordinary process. As the door is about the height of his chin, the labor of doing the work is very greatly increased without any apparent corresponding advantage. The process does not save any fuel. The labor is difficult, and its only advantages seem to be the removal of the emery, which, as it sometimes contains considerable phosphorus, injures the quality of the iron, and the elimination of a part of the sulphur of the ore in the retorts. It has not come into very general use. At the works in Ticonderoga fine ore is brought by a simple piece of machinery and regularly discharged into the nozzle, so that it is constantly blown into the furnace. It is thought that a saving of ore and fuel is effected in this way, but no results have as yet been

shown to prove it. At the same works the reheating was done in an ordinary reheating furnace, which did not seem to be working very well, so that the steam hammers which were used did not have any special advantage, as the loupes were usually too cold.

This bloomary process is destined to disappear. It is only applicable to very cheap, pure, and very poor ores, which can be enriched by dressing, but which cannot be profitably sold for other purposes, where transportation is very difficult, labor cheap, and capital small, and where the demand for this kind of iron is great enough to give an exceptional profit for the product. In some localities it has been kept alive, when it yielded no profit, by the store system, the store yielding the profit, and the men accepting a reduction of wages rather than have the works stop. As it is expensive in labor and fuel, and the output is small, it must disappear whenever transportation becomes easy, except possibly in a few localities, like Crown Point, where a larger quantity of poor, pure, fine ore is produced than would be safe to use in the blast furnace, in which case the ore made use of in this process would have to be thrown away, or left to accumulate. Under such conditions, the ore being counted at a very low price in a country where charcoal can be cheaply made, the process may be used; but it is destined, at no distant day, to disappear before the open-hearth steel, which can be made of high quality and in large quantities. Its only advantage appears to be a removal of the phosphorus, and, since it is not homogeneous, the iron made has to be remelted either in the crucible or open hearth to make a marketable article, with the exception of a very small amount used for nails and the coarser variety of tools.

In conclusion I wish to express my thanks to Mr. G. Chahoon, of the J. & J. Rogers Iron Co., and to Mr. W. H. Case, one of our members, who have assisted me in getting exact drawings and details of this very interesting process.

*THE COST OF MILLING SILVER ORES IN UTAH AND NEVADA.**

BY R. P. ROTHWELL, M.E., NEW YORK CITY.

THE milling of silver ores has arrived at a great degree of perfection in the mining districts of our Western States and Territories, and I have thought the record of the practical results obtained at the present time, in a few of the principal districts, would prove of value and interest to many of our members, both at home and abroad.

An American silver mill is composed, in descending order from where the ore is dumped, of:

1st. A rock-breaker, usually of the Blake or Alden pattern;

2d. The battery of stamps, the number of which varies from 5 to 120, generally in batteries of 5 or 10 heads, so arranged that a single battery may stand while the others are running;

3d. The tanks where the crushed ore settles (if it is a wet stamping-mill), or of a drying-floor if the crushing is done dry;

4th. Amalgamating pans;

5th. Settling pans;

6th. Agitators or pans in which mercury escaping in the tailings is caught;

7th. Blanket sluices, over which the tailings run, and where mercury and unamalgamated sulphides are saved. A reference to the engravings (see plate), which show one of the latest forms of an 80-stamp silver-mill run by a turbine water-wheel, will show clearly the arrangement of the several parts.

THE ONTARIO, UTAH, SILVER MILL.

One of the best examples of a silver-mill, and at the same time one of the most successful concerns in the West, is the mill of the Ontario Silver Mining Company, near Salt Lake, in Utah. This mill treats the ore from the Ontario mine, ore which at present is very base, being composed of zinc, lead, and silver sulphides and silver chloride in a quartz gangue. This ore has become baser as the mine attained greater depth, though the vein holds its own, or rather increases, both in thickness and richness. The mill is managed in a skilful and economical manner, though the necessity for

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* Read at the Montreal meeting, September, 1879.

roasting the pulverized ores in Stetefeldt furnaces necessarily makes the cost of milling much greater than in the case of such free milling ores as those of the Comstock mines, and especially than those unrivalled free milling ores of the silver-bearing sandstone district of Silver Reef, in Southern Utah.

I will here summarize briefly the various operations through which the ore goes in the Ontario mill, giving at the same time the number of men employed in each part of the work, and the wages they receive, as well as the quantity of chemicals used. For this data I am indebted to the courtesy of the very efficient general manager of the Ontario Company, Mr. J. C. Chambers, and to the skilful young superintendent of the Ontario mill, Mr. J. E. Gallagher.

The mill is charged with the hauling of the ore from the mine to the mill, which costs by contract fifty cents per ton dry. The moisture averages about eleven per cent. The distance is about a mile, down grade all the way, four-horse teams hauling six to seven tons at a load. The ore, which is weighed by one weigher at \$4.00 a day, is delivered on the ore floor screened, by passing over iron bars $2'' \times \frac{3}{8}'' \times 9'$ long, with spaces of two inches between; the inclination of the bars is about 30°. The coarse ore goes through two Blake crushers, attended by one man at \$2.50 per day, and is crushed to the size of a pigeon's-egg, and then goes along with the fine ore which has passed between the screen bars through shoots to the drying-floor, which is heated by thirteen flues, seven of which are heated by auxiliary fires and six by the waste gases from the Stetefeldt furnaces and one auxiliary fire. It was at first supposed that the waste gases from these roasting furnaces would have sufficed for the drying-floor, but experience has shown that they do not. From the shoot the ore is carried over the drying-floor in cars, which carry 1000 pounds of ore (dry), and from this measure the quantity of salt is gauged. The ore is spread over the floor to a depth of three inches, and, after drying for two hours, 17½ per cent. wet (about 15 per cent. dry) of salt is scattered over the ore. The salt used is evaporated from the waters of Salt Lake, and costs at the Ontario mill about \$7.00 per ton, \$5.50 of this being for hauling from the salines to the mill. The ore is left from one and a half to two hours after the addition of the salt for the purpose of drying the same, and is then turned with shovels. This work is very injurious to the men, who quickly get swelling of the legs, it is supposed from arsenic in the ore, and in this, as in most other trying occupations, it is found that indulgence in alcoholic stimulants

quickly incapacitates the workmen. Three men on a shift and three shifts a day, with wages at \$3.50 per day, are required for the work on the drying-floor. After turning, the ore and salt remains on the floor for from one-half to one and a half hours, and is then scraped off by a mechanical scraper to the self-feeder of the battery.

The battery consists of forty 800-pound stamps, which drop 8 to $8\frac{1}{2}$ inches 92 times per minute. The shoes, dies, and tappets are of steel, cast in Collinsville, near Hartford, Conn. The dies and shoes wear from two to six and sometimes eight months; the tappets and cams last twelve months. The cams weigh about 250 pounds, and the iron stamp stems are $3\frac{1}{8}$ inches diameter and 14 feet long. The stamp screens are of brass wire, No. 30, that is, 900 meshes in a square inch. Formerly a No. 50 screen was used, but as the ore became more base it would not splash high enough on the screen, and the amount it was possible to mill dry became greatly reduced. Mr. Gallagher tried the larger mesh screens with great success, it being found that the roasting and chlorination in the Stetefeldt furnaces is quite as perfect, if not more so, on the coarse ore as on the fine. The battery requires the attendance of one man each shift; wages \$4.00, and three shifts a day. The crushed ore is conveyed from the battery by a screw shaft, working in a box parallel with the battery to a common flour-mill elevator with Russia-iron cups 22 inches apart, on a rubber belt 6 inches wide, and which lasts two years. This raises it to a vertical height of about 50 feet, to the pockets over the Stetefeldt furnaces; thence it passes through two screens, the bottom one of which is fixed, and stands on the top of the water-jacket, and the upper one vibrates, making about fifty strokes per minute. It is run by cone pulleys, and over the vibratory screen are four brass bars called "agitators," which keep the ore moving on the screen. The screens are of No. 18 steel plate, punched with ten holes to the inch, and the ore sifts down through them into the furnace. This is 46′ 6″ high, and from the bottom the ore is drawn every three-quarters of an hour. One man on a shift (wages \$4.00), and three shifts a day, is all the labor required at the screens. This man takes samples for assay every hour.

The accompanying illustration and description of the Stetefeldt furnace, as in use at the Ontario mill, will be of interest:

Of the drawings, Figure 1 represents a vertical section of the Stetefeldt furnace, showing its latest and most improved mode of construction, and Figure 2 is a sketch of the Stetefeldt feeder:

DESCRIPTION OF THE STETEFELDT FURNACE.

A is the shaft into which the pulverized ore is showered by the feeding-machine placed on the top of the cast-iron frame B. The shaft is heated by two fireplaces (C). The ash-pits of these are closed by iron doors, having an opening (E), provided with a slide, so that more or less air can be admitted below the grate, and, consequently, more or less heat generated. In order to obtain a perfect combustion of the gases, leaving the firebox through the slit (T), an air-slit (U), connected with the air-channel (F), is arranged above the arch of the firebox. This slit also supplies the air necessary for the oxidation of the sulphur and the base metals. Another advantage of this construction is, that the arches above the firebox and firebridge are cooled and prevented from burning out. The roasted ore accumulates in the hopper (K), and is discharged into an iron car by pulling the damper (L), which rests on brackets with friction rollers (M). N is an observation door, and also serves for cleaning the firebridges. O are doors to admit tools in case the roasted ore is sticky and adheres to the walls. The gases and fine ore-dust, which forms a considerable portion of the charge, leave the shaft through the flue (G). The doors (R) are provided to clean this flue, which is necessary with some ores about once a month. D is an auxiliary fireplace, constructed in the same manner as the fireplaces on the shaft, which is provided to roast the ore-dust escaping through the flue (G) in passing through the chamber (H). P are doors for observation and cleaning. The larger portion of the roasted dust settles in the chamber (V), provided with discharge hoppers (I), from which the charge is drawn into iron cars by moving the dampers (S). The rest of the dust is collected in a system of dust-chambers (Q), connected with a chimney, which should rise from forty to fifty feet above the top of the shaft. At the end of the dust-chambers is a damper by which the draught of the furnace can be regulated. The dry kiln can also be used as a dust-chamber, and the waste heat of the furnace utilized for drying the ore before crushing it. The firing of the furnace is done on one side, and all discharges are located on the opposite side.

Description of the Stetefeldt Feeding Machine.—The feeding machine is shown in Fig. 2. The cast-iron frame (A), which is placed on top of the shaft, is provided with a damper (B), which is drawn out when the furnace is in operation, but inserted when the feeding machine stops for any length of time, or if screens have to be replaced. C is a cast-iron grate, to the top of which is fastened

the punched screen (D). The latter is made of Russia sheet-iron, or of cast-steel plate, with holes of one-eighth to one-tenth of an inch in diameter. Above the punched screen is placed a frame (E), to the bottom of which is fastened a coarse wire screen (F), generally No. 3, made of extra heavy iron wire. The frame (E) rests upon friction rollers (G). The brackets (H) which hold the friction rollers can be raised or lowered by set-screws, so that the wire screen (F) can be brought more or less close to the punched screen (D). The brackets (K) carry an eccentric shaft (L), connected with the shaft (M), from which the frame (E) receives an oscillating motion. To the brackets (N) are fastened transverse stationary blades (O), which come nearly in contact with the wire screen (F), and can be raised or lowered by the nuts (P). These blades keep the pulp in place when the frame (E) is in motion, and also act as distributors of the pulp over the whole surface of the screen. The hopper (I) receives the ore from an elevator which draws its supply from a hopper into which the pulverized ore is discharged from the crushing machinery. The ore is generally pulverized through a No. 40 screen. By means of a set of cone pulleys the speed of the frame (E) can be changed from twenty to sixty strokes per minute, whereby the amount of ore fed into the furnace is regulated. This can also be done, to some extent, by changing the distances between the punched screen (D), the wire screen (F), and the blades (O).

The arrangement of the feeding and conveying machinery has been lately much improved and simplified, so that no heavy and large building is required on top of the furnace, and the fireman can easily regulate the supply of ore to the feeding machine, and keep the same in running order.

The furnaces have 13 dust-chambers, 8 and 10 feet high, by 8 feet long, by 4 and 5 feet wide, which collect about 25 tons of dust per month; this averages a little higher grade than the ore; about 56 per cent. of it is soluble. The two furnaces have a capacity of 50 to 55 tons a day of the ore now being treated, and consume daily 9½ to 10 cords of wood, which costs $4 per cord. Firing requires 1 man (wages $4) on each shift, three shifts per day.

After leaving the furnace the ore goes to the cooling-floor, where it remains piled up for 18 hours; this assists the chlorination about 8 per cent. on the ore from the old furnace, and 3 per cent. on that from the new. After this it is wet with water and run in cars to the pan-room. Three shifts, of two men per shift, at $4 a day wages, attend to the cooling-floor; two shifts, with 2 men on a shift,

wages $4 a day, attend to the cars, and take samples for assay from every car.

AMALGAMATION IN PANS.

There are 24 pans, which are charged each with 2500 pounds (dry) of pulp and about 1 per cent. of salt, and the pulp is made by the addition of hot water into a paste of about the consistency of thin mortar. The muller is held about 1 inch off the bottom, and consequently does not grind in the pan. It makes about 65 revolutions per minute and runs for 8 hours. About 1 pound of zinc, costing 9 or 10 cents per pound, and 300 pounds of mercury are added after the pan has run 1 hour and is hot. The labor expended is 2 shifts, of 2 amalgamators on each, at $4.50 per day wages. The loss in mercury is about 3 pounds per ton.

From the pans the pulp is drawn into the 12 settlers, which run 4 hours, making 40 revolutions per minute. After running 1 hour, cold water is let run in and overflow, carrying off the tailings, samples of which are taken for assay.

The amalgamation proceeds, very rapidly at first, in the pans. About half of all that is obtained is amalgamated in the first $1\frac{1}{2}$ hours; at the end of three hours $\frac{2}{3}$ is amalgamated, and after six hours 85 per cent. of what the mill is working to is in the form of amalgam. Nothing material is gained by running the pans beyond eight hours. The mill is working up to from 88 per cent. to 92 per cent. of the assay value of the ore, that being counted as the amount chlorinated. The tailings carry from 12 per cent. to 8 per cent.

The amalgam is strained in canvas bags and sent to the retort, the charge for which is 2000 pounds and the time required $7\frac{1}{2}$ hours. The fuel used is charcoal, about 8 bushels at 20 cents per bushel ($12\frac{1}{2}$ pounds) being required to melt 1 bar of about 1400 ounces. There is 1 retorter and 1 melter working one shift a day, wages $4. The bullion obtained runs about 600 fine and contains no gold. The average grade of ore treated is from $100 to $130 per ton, and the amount treated from 50 to 55 tons per day; when the ore was less base the mill treated 65 tons per day. The salaries charged against the mill are those of the mill superintendent and the assayer. The fuel consumed in the mill is about fifteen cords per day, including that used in the two roasting furnaces.

The following table gives the average cost per ton in labor and material for treating Ontario ore. These figures are kindly furnished me by the superintendent of the company:

ACTUAL RUNNING EXPENSES OF WORKING ONTARIO ORE, ESTIMATED FROM A PRODUCTION OF FIFTY TONS PER DAY.

LABOR.

No. men.	Occupation.		Per day.	Per ton.	
1	Foreman,		$10.00	20	
1	Assayer,		6.00	12	
3	Machinists,	at $4 00	12.00	24	
2	Carpenters,	" 4 00	8 00	16	
2	Blacksmiths,	" 4 00	8.00	16	
2	Engineers,	" 4.00	8 00	16	
2	Foremen,	" 3.50	7.00	14	
9	Dry floor,	" 3.50	31 50	63	
3	Battery,	" 4.00	12.00	24	
6	Roasters,	" 4.00	24.00	48	
12	Cooling floor,	" 4 00	48 00	96	
4	Carmen,	" 4 00	16.00	32	
4	Amalgamators,	" 4.50	18.00	36	
1	Retorter,	" 4.00	4 00		
1	Melter,	" 4 00	4 00	16	
4	Laborers,	" 2.50	10.00	20	
4	Watchmen,	" 3.00	12.00	24	
2	Ore floor,	" 3.50	7.00	14	
3	Clerks,	" 4.00	12.00	24	
66			$257.50	$5 15	$5.15

SUPPLIES.

Salt, 10 tons,	at $8.00	$80.00	$1.60	
Quicksilver, 175 lbs.,	" .50	87.50	1.75	
Wood, 15 cords,	" 4 50	67.50	} 3.53	
Coal, 12 tons,	8.25	99.00		
Castings,			1.50	
Oil and waste,			25	
Sundries, chemicals, etc.,			50	
Hauling from mine,			49	
Charcoal, assaying, and melting,			25	
			$9.87	$9.87
				$15.02

The above items, aggregating $15.02, are not, however, the total expense of milling; there is general superintendence, office expenses, repairs, insurance, royalty on use of roasting furnace, etc., to be added to this.

COST OF MILLING SILVER REEF ORES.

The Ontario ores are probably as expensive to mill as any silver ores in the country, the head of stamps crushing but about 2¼ tons per 24 hours, and the ores require an expensive roasting. I will now cite an example of what are probably the easiest milling silver

ores in this country, namely, those of Silver Reef, Utah. These ores are silver-chloride impregnations of sedimentary sandstones, which crush so easily that a 750-pound stamp will crush through a 40-mesh screen an average of from seven to eight tons per 24 hours. The mills in the Silver Reef district are all small, containing only 5 and 10 stamps, and though they contain as much as 12 pans, with a capacity of $1\frac{1}{2}$ tons to the head of stamps, the battery capacity is in all cases greater than that of the pans. Some of the mills have averaged the month through $8\frac{1}{2}$ tons to the head of stamps; nor does that appear to be a maximum limit. The ore under the stamp disintegrates and passes rapidly through the screens as fine sand; indeed it seems probable that Cornish rolls would crush this ore fine enough for amalgamation, and would do so with wonderful rapidity. As the ore is also remarkably pure, no impurity except a little copper, which occurs in a few of the mines, is found in the bullion. The cost for chemicals is also extremely low. Considering the fact that the mills are so small, and that some of the items are therefore necessarily high, the cost of milling is the lowest of any silver ores in this country:

	Christy M. and M. Co.		Stormont Co.		Leeds Co. in 1878.	Leeds Co. 4 mos. 1879.
Per ton of 2000 lbs.	14,249 tons.		9983 tons.		12,064 tons.	4679 tons.
Labor and salaries,		$2.85		$2.97	$2.20	
Bluestone, . . .	2.1 lb.,	31	$1\frac{3}{4}$ lb.,	26		
Mercury, . . .	1.22 "	58	1.13 "	57		
Salt, ...	25.8 "	51	20 "	$29\frac{1}{2}$	3.22	
Fuel, ...		1.31	*	$45\frac{1}{2}$		
General Supplies,		87		45		
Incidentals, . .		41		12		
		$6.84		$5.12	$5.42	$4.12
Hauling, . . .		73		2.00	32	25

The tailings vary greatly in richness according to the character of the ore milled. From sandstone ore they will carry $3 per ton, while from the shale ore they may run $10 or more.

MILLING COMSTOCK ORES.

There has always been a great difficulty, not to say impossibility, in getting reliable figures as to the cost of milling Comstock ores. The reason for this difficulty is easily understood when we say that the mills belong to a private company composed of the members of

* Stormont mill is driven by water-power.

the Bonanza firm, who also control the management of the mines. The managers of the mines pay the mills high mill charges, formerly about $20, now $11 per ton of stuff milled, and the mill to own the tailings. Formerly the mill contracted to return the mine not less than 65 per cent. of the assay value of the ore; but as the assayers were in the employ of the owners of the mills this guarantee was as easily complied with as it was worthless. Now I am told by Mr. Patton, the Bonanza firm's general superintendent, that the mills return "all they get out, which is about 72 per cent.," the mill still owning the tailings. I think the percentage here stated is open to question; nevertheless, since no opportunity is afforded for its verification, we must accept it for what it is worth.

The following particulars, though not taken from the books and not "official," are still, I believe, so near the inaccessible "bottom facts" that they may be of interest to some of our members. The data were collected with care, both during a visit to the mill and afterwards from well-informed sources.

The California mill, Virginia City, is one of the finest in the world and is admirably managed; it contains 80 stamps, which fall 7 to 8 inches, and make 90 to 100 drops per minute. The ores are very easily crushed, and the full capacity of the mill is estimated at 360 tons per 24 hours, or 4½ tons per head of stamps per day; probably the average work is 320 tons a day, or 4 tons per stamp; the stamping is done wet. The stamps are fed automatically, and the total number of men employed in the stamp-mill per 24 hours is 12, namely, 2 engineers, 4 feeders, 2 firemen, 2 repairers, and 2 binmen.

From the stamps the ore runs through a trough to the pan-mill, situated at some distance down the cañon. In this there are 40 pans running on ore and 4 pans on tailings. The average charge of pans is 3600 pounds of ore worked in 5 hours; the mullers run about 90 revolutions, and bear on the bottom, or grind. From 12 to 24 pounds of salt, and about one-third as much bluestone (sulphate of copper) is added to the charge, and after 3 hours' grinding 350 pounds of mercury is added. These quantities vary with the richness and character of the ore. The average loss in mercury is from 2 to 2½ pounds per ton of ore. The settlers (20) run about 18 revolutions and 2½ hours, and their tailings run into the agitators and thence over the blanket sluices. The labor in the pan-mill per 24 hours is 6 amalgamators, 20 tank shovellers, 2 pan shoers, 2 amalgam-men, 2 engineers, 2 firemen, 5 woodmen, 3 repairers, 3 oilers, 1 re-

torter, 1 extra roustabout, 1 lamp-boy, 2 night watchmen, 2 foremen, 1 superintendent—total 53 men.

The tailings form about 10 per cent. of the ore and are reworked in the tailing-mill, and then run over blanket sluices; this requires 2 amalgamators, 4 shovellers, 1 cartman, 7 blanket-men, and 1 boss blanket-man—15 men. Total in stamp, pan, and tailing mills, 80 men per 24 hours; the wages are from $3 to $4 per day. The fuel consumed per 24 hours in the battery-mill (360 tons) is 80 to 90 cords, and in the pan and tailing mills (369 tons) 38 to 40 cords; total 118 to 130 cords at $10 per cord.

There are two sets of blanket sluices, each of 6 tables, 300 feet long, with a grade of about $2\frac{1}{2}°$. In the first set $\frac{5}{8}$ of all the tailings recovered is caught, and $\frac{3}{8}$ in the second set. About 8 per cent. of the stamped product is saved in these sulphurets, which assay sometimes $20. These blanket tailings are mixed with from 3 to 10 pounds of salt per ton, exposed to the air, and then mixed with the tailings of the settlers in the agitators, and the mixture averages, I am informed, about $7 per ton.

THE EIGHTY-TON STEAM-HAMMER AT CREUSOT.

BY J. A. HERRICK, M.E., PITTSBURGH, PA.

FOR a long time, especially in Europe, heavy pieces of forgings, such as cannon, armor plates, marine shafting, etc., have been steadily augmented in size, more particularly since steel has been substituted for wrought iron. To keep pace with these forgings several very large hammers have been, from time to time, erected.

The most powerful and thoroughly appointed of all now in operation is that designed and built by Messrs. Schneider & Co. at their works at Creusot, and it is the design of this communication to give to the members of the Institute a concise description of this remarkable piece of mechanism, which I have taken from an article published in the *Annales Industrielles*.

This hammer is rated nominally at 80-tons power, but its capacity is, in reality, far greater, as will be seen by the comparative figures given later. The complete plant, comprising the hammer, the cranes, the furnaces, and the building which shelters the whole, is distin-

guished by breadth of conception and accuracy of detail, whereby not only harmonized proportions are insured, but also the greatest economy and convenience in actual practice.

We will consider, 1st, the site, the building, etc.; 2d, the hammer; 3d, the four cranes; 4th, the four furnaces; and, 5th, the accessories.

1st. *The Site and the Building.*—The framework of the building, which shelters the hammer, cranes, and furnaces, is entirely of metal, a necessary precaution against fire. The building is 50 meters (164 feet) long, 35 meters (114 feet 10 inches) wide, and 17 meters (55 feet 9¼ inches) high below the trusses. The height between the ground and the ridge-pole is 25.5 meters (83 feet 8 inches), and the total height between the ground and the upper part of the ventilator is 28.3 meters (92 feet 10 inches). The ground covered is 1750 square meters (18,832⅓ square feet).

A service bridge, resting upon the beams forming the wall-plates, serves to support two winches of a power of twenty tons each, which are designed to manipulate the necessary parts of the hammer in case of repairs.

The 80-ton hammer occupies the centre of the building. It is served upon each face, before and behind, by two cranes. One of these cranes is of 160 tons power; the other three are each of 100 tons power.

The four furnaces, symmetrically arranged in relation to the centre of the building, and the faces of the hammer, are placed obliquely, to facilitate the handling of the ingots when the cranes take them to conduct them to the hammer or remove them after forging.

A railroad of 1.44 meters (57 inches) between rails, placed in the centre of the great hall of the large building, is expanded like a V at its entrance into the central building, and each branch of the V conducts to a crane. This road is used to bring the ingots, cast in a special building, to the hammer, and also to convey the forged pieces to the fitting shops.

2d. *The Steam-hammer.*—The steam-hammer is composed of four distinct parts, the foundations or substructure, the legs, with their entablature, forming the superstructure, the steam-cylinder, with its valves and connections, and finally, the active mass, that is to say, the ram, with its rod and die.

The foundations are composed of a solid mass of masonry laid in cement, resting upon the bed-rock, which is here found at a depth of 11 meters (36 feet 1 inch) below the soil, of an anvil block of

cast iron, and a filling-in, composed of oak wood, to diminish, by its elasticity, the transmission of the vibrations resulting from the blows of the hammer. The solid mass in masonry consists of a cube of 600 meters ($1968\frac{1}{2}$ cubic feet). Its upper face is covered with a bed of joists of oak wood of about 1 meter (3 feet $3\frac{3}{8}$ inches) in thickness, placed horizontally, upon which rests the anvil block. At Perm it was found more convenient to make the anvil block in a single piece, and to mould and cast it on the very spot which it was destined to occupy. The weight of this block (Perm 50-ton hammer) is about 622 tons.

This method being deemed objectionable, and it being impossible to transport to its final site a mass of iron weighing 720 tons, the Messrs. Schneider determined to make the block in six horizontal courses, resting one upon the other, with planed surfaces. Each is formed of two pieces, excepting the upper course supporting the anvil, which consists of a single piece, and weighs 120 tons.

The anvil block, which is 5.6 meters (18 feet $4\frac{1}{2}$ inches) high, and has a surface of 33 square meters (355 square feet 30 square inches) at the base, and 7 square meters (75 square feet 50 square inches) at the top, is thus formed of eleven pieces. The parts of the same course are strongly bound together, and each course is also securely joined to the one above and below. The empty space between the anvil block and the inner side of the pit in which it is placed is filled with joists of oak wook, placed on end and driven home. The legs incline towards each other in the form of a letter A, and are bolted upon a foundation plate, imbedded in the masonry surrounding the anvil block, and are joined at the top by the table. The legs are cast hollow, and the transverse section is rectangular; each leg is in two parts, fastened about the middle by a flange and bolts. The anvil block is independent of the legs. The slides are joined to their respective legs by means of bolts. The legs are strongly bound together by four wrought-iron plates, which at the same time hold the slides. The height of the legs is 10.25 meters (33 feet 8 inches), and they weigh, with the slides, 250 tons. The iron plates binding the legs together weigh about 25 tons, and the foundation plates supporting the legs 90 tons. This arrangement of the legs, bound together by iron plates, is of very great stiffness, as experience has shown since the hammer has been in regular operation.

The weight of the table resting on the legs is 30 tons, and upon it is placed a steam cylinder, composed of two pieces, each 3 meters (9 feet 10 inches) high, fastened with flanges and bolts. The distri-

bution of steam is effected by means of two balanced valves; it is single-acting. The diameter of the cylinder is 1.90 meters (6 feet $2\frac{4}{5}$ths inches) which is equal to a surface of 27,345 square centimeters (4409 square inches), deducting the area of the rod, which is 36 centimeters (158 square inches). With the steam pressure of five atmospheres this gives to the piston a power of about 140 tons. As the weight of the active mass to be raised is 80 tons it is apparent that there is ample power for a sufficiently rapid upward speed of the piston, and that the weight of the active mass can be increased in case of need.

The stroke of the piston is 5 meters (16 feet 5 inches). This fall multiplied by 80,000 kilograms, the weight of the mass, gives the available work of the hammer as 400,000 kilogrammeters. The 50-ton hammer at Krupp's works, in Essen, whose stroke is only 3 meters (9 feet 10 inches), gives a power of 150,000 kilogrammeters. If we suppose the two hammers acting upon an ingot 1.5 meters (4 feet 11 inches) high, the Creusot hammer would still have 280,000 kilogrammeters available, while that at Essen would only have 75,000 kilogrammeters. It is thus apparent that in this case the Creusot hammer has more than three times the power of that at Essen.

The space between the legs is 7.5 meters (24 feet $7\frac{1}{4}$ inches), and the free space under the cross-pieces is 3.2 meters (10 feet 6 inches), leaving plenty of room around the hammer—an indispensable condition for the handling of the enormous blocks. The height of the hammer from the foundation plate to the upper side of the cylinder is 18.6 meters (61 feet). Adding to this dimension 5.6 meters (18 feet $4\frac{1}{2}$ inches), the height of the anvil block, and 6 meters (19 feet 8 inches) for that of the masonry which supports it, we find that this colossal structure is almost 30 meters (99 feet) high. In spite of this unfavorable condition for stability, in spite of the enormous effect resulting from a blow of 400,000 kilogrammeters, the structure has been so well proportioned that it does not vibrate, and the foundations, cushioned by the filling of oak joists, transmits to the ground very feeble vibrations, less marked than those of hammers of far inferior power.

The working of the valves is accomplished by rods attached to their levers at one end, and descending along one of the legs to a platform carried by this leg, at a height of about 3 meters (9 feet 10 inches) above the ground. The workman in charge of the valves stands on this platform, and is thus protected from the intense heat

radiated by the metal during forging, and from the hot flying scale and cinder.

3d. *The Cranes.*—The four cranes serving the hammer are of the same type; they differ from one another only in power. As we have said above, three cranes have a power of 100 tons each, and one of 160 tons. They belong to a class of cranes with a single lower pivot (not hung from roof or other support), and have the form of a neck of a swan. These cranes are made of sheet and T iron; they turn upon their pivot, while supporting themselves at the ground level by a hoop sliding vertically. This hoop forms part of a sort of cast iron tubbing, strongly imbedded in the masonry of the foundations; and the tubbing is bound to the cast-iron plate which supports the legs of the hammer. As this arrangement applies to each of the four cranes, their foundations are thus united with one another and with that of the hammer over a large space, giving to the whole great stability. From the pivot to the top of the jib, the height of each crane is 17.40 meters (57 feet 1 inch), which is thus divided: From the pivot to the ground line 8.4 meters (27 feet 6⅜ inches), and from the ground to the top of the jib 9 meters (29 feet 6⅓ inches). The radius of the circle of revolution of the cranes is 9.35 meters (30 feet 8 inches). Each of these cranes possesses four movements, imparted by a little steam-engine carried by the crane itself, and which can develop 60-horse power, with a speed of 250 revolutions per minute.

These movements are: 1st. A circular movement of the crane upon its axis. 2d. A movement for the elevation of the load. 3d. A movement for the horizontal transfer of the load. 4th. A movement for the rotation of the load in the chains. The first three have nothing of special interest; they are accomplished by means of gears and connections, permitting the direction to be readily changed. The load is attached to a cap suspended from a system of movable pulleys, over which passes a chain coiled about a drum, with a helicoidal groove, fixed to the main part of the crane. The horizontal movement of the load is effected by a little carriage running upon rails, placed upon the arm of the crane, and moved by a rack and pinion system in connection with the engine by means of chain belts. The most novel movement is that of rotation of the piece. This rotation in hammers of small power is accomplished by hand-power, with the aid of levers and a large number of men, but this method would not be practicable in case of pieces of 100 tons and more, which this hammer is intended to forge.

The apparatus employed in the Creusot cranes, and which is already in use in other works, is composed of a hollow shaft, in two sliding parts, supported upon the body of the crane, ending in the cap, to which is attached the piece to be forged. The two extremities of the shaft carry each a universal joint (*à la Cardan*), which permits it to follow the vertical movement of ascent and descent of the piece, as well as the oscillations of the same, while its hollow telescopic arrangement enables it to assume a variable length adapted to the horizontal motion of the load. The extremity of the shaft placed in the cap transmits the movement that it has received from the side of the crane to a series of retarded gears, which set in motion a pulley, which, in its turn, operates the chain passing around the ingot to be forged.

The arrangements which have just been described apply to the 160-ton crane, as well as those of 100 tons; the only difference between them is in the dimensions, which are in direct proportion to the power of the cranes. The total weight of the 100-ton cranes is 110 tons; that of the 160-ton crane is 140 tons.

The workman charged with the manipulation is placed upon a small platform attached to the crane, in front of the steam-engine. He has within his reach a system of levers, which enables him to direct the movements according to the order he receives from the foreman.

4th. *The Furnaces.*—The furnaces each occupy a space of 7.8 meters (25 feet 7 inches) by 3.60 meters (11 feet 9$\frac{3}{4}$ inches),* and are 10 meters (32 feet 9$\frac{3}{8}$ inches) in total height. The dimensions of the interior are 4.30 meters (14 feet 1$\frac{1}{3}$ inches), by 3.4 meters (11 feet 2 inches), with a height of 2.6 meters (8 feet 6$\frac{1}{3}$ inches) under the arch. The opening through which the pieces are introduced into the furnace is 3.5 meters long (11 feet 5$\frac{3}{4}$ inches), by 2.3 meters (7 feet 6$\frac{1}{2}$ inches) high; the door which closes it is operated by a hydraulic apparatus working upon chains. The Siemens generators supplying the gas for the four furnaces for the 80-ton hammer, as well as for the other furnaces for the tire and forging shop, are 36 in number, forming a battery of nine groups of 4 producers each; they are situated some distance back of the shop.

The very complete engravings of the whole plant on the accom-

* The dimension should probably be 3.9 meters (12 feet 9$\frac{1}{2}$ inches), since the interior is given as 3.4 meters, which would only allow 4 inches of brick to each side of the furnace.

panying plates will facilitate the understanding of the description which has just been given.

5th. *The Accessories.*—A plant of this sort necessarily involves a complete secondary plant of machine shops, etc., which are of equal importance. Thus, special carriages are needed to convey the ingots to the hammer or pieces to the fitting shop. Certain steel forgings, cannon for instance, must be submitted to the operations of tempering and annealing, or it may be necessary to store the completed pieces until it is convenient to ship them. For these handlings there has been constructed outside the building containing the hammer a road of 11 meters (36 feet 7 inches) between rails, upon which travels a rolling bridge, entirely of metal, carrying winches of 100-tons power.

The works of Messrs. Schneider already have six Bessemer convertors of eight to ten tons capacity, eight Siemens-Martin furnaces, and two rotary furnaces. All these together are capable of producing the quantity of steel in a state of fusion at one time, so as to be collected in ladles and poured into an ingot mould in a single piece of 120 tons.

To cast and manipulate these immense ingots it has been necessary to dig very deep pits to receive the ingot moulds, to construct ingot moulds capable of holding 120 tons of steel, ladles to receive this metal when melted, and also a crane of corresponding capacity.

The entire plant for the casting and forging of large masses of steel, from the casting of the ingots to the final treatment of the forged pieces, has involved an outlay of not less than three millions of francs. The conception of the plan and the construction of the details reflect the greatest honor on the Messrs. Schneider and their engineers.

For convenience in studying the plates a review of the principal dimensions of the plant is subjoined:

THE PRINCIPAL DIMENSIONS OF THE ENTIRE PLANT.

STEAM HAMMER.

Weight of the active mass, 80 tons.*

* The metric ton is equal to 2204 English pounds, and has been taken as equivalent to the English ton.

Dimensions of the Construction Above Ground.

	French Measures.	English Measures.
Maximum fall,	5 m.	16 ft. 5 in.
Diameter of the cylinder,	1.9 m.	6 ft. $2\frac{4}{5}$ in.
Steam pressure,	5 kilos per sq. cm	71 lbs. per sq. in.
Steam power under the piston,	140 tons.	140 tons.
Diameter of the inlet valve,	0.34 m.	$13\frac{2}{5}$ in.
" outlet " (exhaust),	0.46 m.	$16\frac{3}{10}$ in.
" piston rod (steel),	0.36 m.	$14\frac{2}{10}$ in.
Distance between the slides or width of ram,	1.9 m.	6 ft. $2\frac{4}{5}$ in.
Free space between the legs,	7.5 m.	24 ft. $7\frac{1}{4}$ in.
Height under the legs,	3.2 m.	10 ft. 6 in.
Length of base,	12.6 m.	41 ft. 4 in.
Width of base,	6 m.	19 ft. 8 in.
Height of legs,	10.25 m.	33 ft. 8 in.
Height of steam cylinder,	6 m.	19 ft. 8 in.
Total height from foundation plate to the top of steam cylinder,	18.6 m.	61 ft.

Dimensions Below Ground.

	French Measures.	English Measures.
Height of anvil block,	5.6 m.	18 ft $4\frac{1}{2}$ in.
Area at the base,	33 sq. m.	355 sq. ft. 30 sq. in.
Area at the top,	7 sq. m.	75 sq. ft. 50 sq. in.
Number of layers of block,	6	
Number of pieces in each layer,	2	
(The upper layer is in one piece.)		
Thickness of masonry under the anvil block,	4 m.	13 ft. $1\frac{1}{2}$ in.

Weight of the Superstructure.

	English Measures.
Piston, rod, ram, and die (active mass),	80 tons (2204 lbs).
Cylinder,	22 tons.
Table,	30 tons.
Legs and slides,	250 tons.
Cross-bar plates of the legs,	25 tons.
Foundation plates,	90 tons.
Accessory pieces,	35 tons.
Total weight of part of the construction aboveground,	532 tons.
Anvil block and anvil,	750 tons.
Total weight of the construction,	1282 tons.

CRANES.

	French Measures.	English Measures.
Three cranes of the power of,		100 tons.
One crane " "		160 tons.
Radius of revolution of the four cranes,	9.35 m.	30 ft. 8 in.
Height between the ground level and upper part of the jib,	9 m.	29 ft. $6\frac{1}{3}$ in.

	French Measures.	English Measures.
Height between the top of the pivot and the ground level,	8.4 m.	27 ft. $6\frac{3}{8}$ in.
Total height of each crane,	17.4 m.	57 ft. 1 in.
Motive power of each crane,		60-horse-power.
Diameter of the cylinders,	0.26 m.	$10\frac{1}{4}$ in.
Length of stroke,	0.3 m.	12 in.
Number of revolutions per minute,		250
Weight of the metallic part of crane of 100 tons,		110 tons.
" " " " 160 tons,		140 tons.

FURNACES.

Exterior.

	French Measures.	English Measures.
Length over all,	7.8 m.	25 ft. 7 in.
Width over all,	3.6 m.	12 ft. 8 in.
Total height,	10 m.	32 ft. $9\frac{3}{8}$ in.

Interior.

	French Measures.	English Measures.
Length,	4.3 m.	14 ft. $1\frac{1}{4}$ in.
Width,	3.4 m.	11 ft. 2 in.
Height under the roof,	2.6 m.	8 ft. $6\frac{1}{3}$ in.

Charging Door.

	French Measures.	English Measures.
Length,	3.5 m.	11 ft. $5\frac{3}{4}$ in.
Height,	2.3 m.	7 ft. $6\frac{1}{2}$ in.

IRON BUILDING.

	French Measures.	English Measures.
Length,	50 m.	164 ft. $0\frac{1}{2}$ in.
Width,	35 m.	114 ft. 10 in.
Height between the ground and under side of trusses,	17 m.	55 ft. $9\frac{1}{4}$ in.
Height between the ground and top of ridge-pole,	25.5 m.	83 ft. 8 in.
Height between the ground and upper part of ventilator, total height over all,	28.3 m.	92 ft. 10 in.

THE DETERMINATION OF SULPHUR IN SULPHIDES AND IN COAL AND COKE.

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THE use of bromine as an oxidizing agent, particularly for sulphur, has become very general in analysis, replacing the stronger oxidizing acids. The object of this paper is to describe briefly the experience which we have gained with this reagent in the laboratory of Lafayette College in the oxidation of metallic sulphides.

Most of the simple sulphides, as blende, pyrite, etc., when exposed to the combined action of an alkaline hydrate and bromine, and finally to hydrochloric acid, are completely and promptly dissolved. The procedure is as follows: The very finely pulverized mineral is first treated, in a beaker, with a solution of sodium hydrate of a specific gravity of 1.25, and heated; bromine is then cautiously added to supersaturation, and finally hydrochloric acid to acid reaction. If any of the sulphide is not taken up, the same operation may be repeated. It is necessary, however, that the mineral be very finely pulverized. Instead of using the pure bromine, a saturated solution of bromine in potassium bromide may be used with equally good effect.

The process is advantageously simplified by making a saturated solution of bromine in the concentrated alkali. This is done by pouring bromine into a solution of sodium hydrate, of the above given specific gravity, until no more is taken up, and then adding a little of the sodium hydrate solution until the liquid does not give off free bromine. The procedure with this solution is as follows: The pulverized mineral is moistened with, say, 10 cc. of the solution, and heated, then hydrochloric acid added to just acid reaction. Two more additions of the alkaline solution, in amounts of 20 cc. each, are added at intervals of about ten minutes, each addition being followed by hydrochloric acid. The total amount of the alkaline solution (containing the bromine) used is, therefore, 50 cc., and the amount of hydrochloric acid should not exceed that necessary to make the solution acid after each addition of the alkali. The mixture should be kept hot. After the final addition of acid, the contents of the beaker is taken to dryness and heated in an air-bath to 110° to 115°C., to render silica insoluble. The dry mass is then taken up by hydrochloric acid and water and, after filtration, the sulphuric acid is pre-

cipitated by barium chloride. In a sample of copper pyrites Mr. F. E. Bachman obtained, in duplicate analyses, 34.05 and 34.12 per cent. sulphur; in zinc blende 32.97 and 33.09 per cent. In another sample of blende Mr. P. W. Shimer obtained 32.71 per cent.

This method of determining sulphur I find especially valuable in the analysis of coal. By the treatment of coal as above described results are obtained which agree very closely. The coal, as such, is not attacked, and the sulphur obtained, therefore, represents that existing in the coal as pyrite, and also as soluble sulphates. The residue left by this treatment has been subjected again to the same process, and yields no more sulphur. On combustion, however, or by complete oxidation, either by oxidizing acids or by fusion, additional sulphur may be obtained, which must represent that combined organically with the coal.

The following are some of the results obtained by Mr. Shimer from bituminous coals by the bromine method. The amount usually taken for analysis was between one and two grams:

Bituminous coals.	Total sulphur by fusion with alkaline carbonates and nitrates.	Sulphur by bromine process.
I.	0.43	0.035
		0.035
		0.035
II.	2.16	1.80
	2.17	1.81
		1.81
		1.83
		1.84
		1.85
		1.87
III.	1.17	0.710
	1.18	0.713
		0.717
IV.	1.48	1.096
	1.49	1.098
	1.50	1.100
		1.100

In comparing the bromine method with others it was found that the treatment with hydrochloric acid and potassium chlorate gave on coals with but little sulphur in the form of pyrite the same results, but on coals with much pyrite the results were decidedly lower than by the bromine method. But too few experiments were tried on this

point to be decisive. The action of nitric acid and potassium chlorate depends upon the nature of the coal. Some coals are converted partly into a brown unmanageable solution, and others are oxidized completely to a clear solution. In the latter case, of course, the total sulphur may be obtained.

As was said above, the sulphur obtained by the bromine method represents both the sulphides and sulphates in the coal. The methods ordinarily given for the separate determination of calcium sulphate are faulty. Sodium carbonate readily attacks pyrite, and dilute hydrochloric acid and even water, when heated for some time in contact with pyrite, with access of air, contain notable quantities of sulphuric acid. It would seem, therefore, necessary to dissolve out the calcium sulphate by means of water with the careful exclusion of air.

The determination of the total sulphur in coal by means of fusion with alkaline carbonates and nitrates, or chlorates, I find unsatisfactory, owing, I think, to the large amount of salts in the solution in which the barium sulphate is precipitated. A much better method is to burn the coal in a platinum boat placed in a glass tube in a current of oxygen. The products of combustion may be absorbed by a solution of bromine in hydrochloric acid, or by a dilute solution of potassium permanganate. The latter, I have satisfied myself, gives equally good results with the bromine. It is absolutely necessary, in this process, as originally pointed out by Muck,* that the combustion-tube should be washed out with water after the completion of the combustion, since sulphuric anhydride condenses in considerable quantity in the tube beyond the boat. It is further necessary, of course, to fuse the residual ash with alkaline carbonates to determine the sulphur which has not been volatilized by the combustion.

I have in progress an interesting investigation on the effect of coking on the sulphur in coal, to determine what influence the nature of the sulphur—whether in combination with iron as pyrite or organically combined with the coal—has on its elimination in coking. These results must, however, be reserved for a future communication.

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* Fresenius, Zeitschrift, xiv, 16.

INDEX.

[AUTHORS' NAMES IN SMALL CAPS; TITLES OF PAPERS IN ITALICS.]

7006